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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXVI.

FEBRUARY, 1896, TO OCTOBER, 1896.

INCLUSIVE.

NEW YORK CITY:

PUBLISHED BY THE INSTITUTE,

AT THE OFFICE OF THE SECRETARY.

1897.

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OFFICERS.

For year ending February, 1897.*

President.

E. G. SPILSBURY, Trenton, N. J.

Vice-Presidents.

WHEATON B. KUNHARDT,	New York City.
JAMES F. LEWIS,	Chicago, Ill.
EDWARD COOPER,†	New York City.

(Term expires February, 1897.)

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ANTON EILERS,	Pueblo, Colo.
CHARLES KIRCHHOFF,	New York City.

(Term expires February, 1898.)

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H. O. HOFMAN,	Boston, Mass.
JOHN A. CHURCH,	New York City.

(Term expires February, 1897.)

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WILLIAM R. WEBSTER,	Philadelphia, Pa.

(Term expires February, 1898.)

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BENJ. SMITH LYMAN,	Philadelphia, Pa.

(Term expires February, 1899.)

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R. W. RAYMOND,
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THEODORE D. RAND,
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* The following officers were elected at the Annual Meeting, February, 1897: *President*, Thomas M. Drown, South Bethlehem, Pa.; *Vice-Presidents* (to serve two years), D. W. Brunton, Aspen, Colo., W. E. C. Eustis, Boston, Mass., James Douglas, New York City; *Managers* (to serve three years), C. W. Goodale, Butte, Mont., Frank Lyman, Brooklyn, N. Y., Frank McM. Stanton, Houghton, Mich.; *Treasurer*, Theodore D. Rand, Philadelphia, Pa.; *Secretary*, Rossiter W. Raymond, New York City.

† Appointed to fill the vacancy caused by the death of Mr. C. A. Stetefeldt.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO FEBRUARY, 1897.

Number.	Place.	Date.	Transactions.	
I.	Wilkes-Barre, Pa.,*	May, 1871,	i.	3
II.	Bethlehem, Pa.,	August, 1871,	i.	10
III.	Troy, N. Y.,	November, 1871,	i.	13
IV.	Philadelphia, Pa.,	February, 1872,	i.	17
V.	New York, N. Y.,*	May, 1872,	i.	20
VI.	Pittsburgh, Pa.,	October, 1872,	i.	25
VII.	Boston, Mass.,	February, 1873,	i.	28
VIII.	Philadelphia, Pa.,*	May, 1873,	ii.	3
IX.	Easton, Pa.,	October, 1873,	ii.	7
X.	New York, N. Y.,	February, 1874,	ii.	11
XI.	St. Louis, Mo.,*	May, 1874,	iii.	3
XII.	Hazleton, Pa.,	October, 1874,	iii.	8
XIII.	New Haven, Conn.,	February, 1875,	iii.	15
XIV.	Dover, N. J.,*	May, 1875,	iv.	3
XV.	Cleveland, O.,	October, 1875,	iv.	9
XVI.	Washington, D. C.,	February, 1876,	iv.	18
XVII.	Philadelphia, Pa.,†	June, 1876,	v.	3
XVIII.	Philadelphia, Pa.,	October, 1876,	v.	19
XIX.	New York, N. Y.,	February, 1877,	v.	27
XX.	Wilkes-Barre, Pa.,*	May, 1877,	vi.	3
XXI.	Amenia, N. Y.,	October, 1877,	vi.	10
XXII.	Philadelphia, Pa.,	February, 1878,	vi.	18
XXIII.	Chattanooga, Tenn.,*	May, 1878,	vii.	3
XXIV.	Lake George, N. Y.,	October, 1878,	vii.	103
XXV.	Baltimore, Md.,*	February, 1879,	vii.	217
XXVI.	Pittsburgh, Pa.,	May, 1879,	viii.	3
XXVII.	Montreal, Canada,	September, 1879,	viii.	121
XXVIII.	New York, N. Y.,*	February, 1880,	viii.	275
XXIX.	Lake Superior, Mich.,	August, 1880,	ix.	1
XXX.	Philadelphia, Pa.,*	February, 1881,	ix.	275
XXXI.	Staunton, Va.,	May, 1881,	x.	1
XXXII.	Harrisburg, Pa.,	October, 1881,	x.	119
XXXIII.	Washington, D. C.,*	February, 1882,	x.	225
XXXIV.	Denver, Col.,	August, 1882,	xi.	1
XXXV.	Boston, Mass.,*	February, 1883,	xi.	217
XXXVI.	Roanoke, Va.,	June, 1883,	xii.	3
XXXVII.	Troy, N. Y.,	October, 1883,	xii.	175
XXXVIII.	Cincinnati, O.,*	February, 1884,	xii.	447
XXXIX.	Chicago, Ill.,	May, 1884,	xiii.	1
XL.	Philadelphia, Pa.,	September, 1884,	xiii.	285
XLI.	New York, N. Y.,*	February, 1885,	xiii.	585

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

Number.	Place.	Date.	Transactions.	
XLII.	Chattanooga, Tenn.,	May, 1885,	xiv.	1
XLIII.	Halifax, N. S.,	September, 1885,	xiv.	307
XLIV.	Pittsburgh, Pa.,*	February, 1886,	xiv.	587
XLV.	Bethlehem, Pa.,	May, 1886,	xv.	lxiii.
XLVI.	St. Louis, Mo.,	October, 1886,	xv.	lxx.
XLVII.	Scranton, Pa.*	February, 1887,	xv.	lxxvii.
XLVIII.	Utah and Montana,	July, 1887,	xvi.	xvii.
XLIX.	Duluth, Minn.,	July, 1887,	xvi.	xxiv.
L.	Boston, Mass.,*	February, 1888,	xvi.	xxviii.
LI.	Birmingham, Ala.,	May, 1888,	xvii.	xix.
LII.	Buffalo, N. Y.,	October, 1888,	xvii.	xxiv.
LIII.	New York, N. Y.,*	February, 1889,	xvii.	xxxi.
LIV.	Colorado,	June, 1889,	xviii.	xvii.
LV.	Ottawa, Canada,	October, 1889,	xviii.	xxiv.
LVI.	Washington, D. C.,*	February, 1890,	xviii.	xxx.
LVII	New York, N. Y.,	September, 1890,	xix.	vii.
LVIII.	New York, N. Y.,*	February, 1891,	xix.	xxv.
LIX.	Cleveland, O.,	June, 1891,	xx.	xvi.
LX.	Glen Summit, Pa.,	October, 1891,	xx.	lxi.
LXI.	Baltimore, Md.,*	February, 1892,	xxi.	xix.
LXII.	Plattsburgh, N. Y,	June, 1892,	xxi.	xxxiii.
LXIII.	Reading, Pa.,	October, 1892,	xxi.	xliv.
LXIV.	Montreal, Canada,*	February, 1893,	xxi.	lii.
LXV.	Chicago, Ill.,	August, 1893,	xxii.	xiii.
LXVI.	Virginia Beach, Va.,*	February, 1894,	xxiv.	xvii.
LXVII.	Bridgeport, Conn.,	October, 1894,	xxiv.	xxxv.
LXVIII.	Florida,†	March, 1895,	xxv.	xix.
LXIX.	Atlanta, Ga.,	October, 1895,	xxv.	xxxiii.
LXX.	Pittsburgh, Pa.,*	February, 1896,	xxvi.	xvii.
LXXI.	Colorado,	September, 1896,	xxvi.	xxix.
LXXII.	Chicago Ill.,*	February, 1897,	xxvii.	

* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida

PUBLICATIONS.

The publications of the Institute comprise:

Pamphlets.

1. The minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed without charge to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price, as follows:

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The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind, by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there is no essential change, the correction of typographical errors and additions of later information being the usual alterations.)

The twenty-six volumes of *Transactions* now published, are for sale as follows:

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All communications and remittances should be addressed to R. W. Raymond, Secretary, P. O. Box 223, New York City.

RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881, 1887, 1890 AND 1896.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote, and members or associates whose post office address shall be outside of the United States, Canada and Mexico shall not be entitled to vote by mail, except upon proposed amendments to the Rules.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance on the first day of each calendar year. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary, and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to

the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members) a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Other meetings shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS AND PUBLICATIONS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute shall

be printed in the Transactions. Intimation, when practical, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers. The published papers and volumes of Transactions shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the Rules.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade; nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute, as provided in these Rules; *Provided*, however, that committees may be appointed by the Council or the Institute to make Investigations and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as Scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these Rules from and after the publication of said announcement by the Secretary.

Proceedings of the Seventieth (Twenty-Sixth Annual) Meeting, Pittsburgh, Pa., February, 1896.

LOCAL COMMITTEES.

Executive Committee.—Jos. D. Weeks, *Chairman;* Geo. H. Clapp, *Secretary;* Julian Kennedy, *Treasurer;* and the chairmen of the following committees:

Reception.—E. M. Ferguson, *Chairman;* Chas. C. Scaife, William H. Singer, William N. Frew, Chas. J. Hilliard, George S. Page, William G. Wilkins, Chas. M. Schwab.

Finance.—Julian Kennedy, *Chairman;* John I. Ricketson, William Metcalf, H. M. Curry.

Hotels and Headquarters.—J. C. McDowell, *Chairman;* W. L. Scaife, J. M. Camp.

Excursions.—Alfred E. Hunt, *Chairman;* Chas. M. Hall, W. L. Scaife, Jas. Darsie, Taylor Allderdice, D. G. Kerr, Theo. Hopke.

Banquet.—William H. Rea, *Chairman;* W. H. Nimick, Horace W. Lash, Taylor Allderdice.

Programme and Badges.—James Gayley, *Chairman;* John F. Wilcox, Hugh Kennedy.

Entertainment of Visiting Ladies.—Mrs. E. M. Ferguson, *Chairman;* Mesdames H. C. Bughman, Geo. H. Clapp, Arthur V. Davis, William N. Frew, James Gayley, Alfred E. Hunt, Julian Kennedy, J. C. McDowell, Geo. S. Page, William H. Rea, Chas. C. Scaife, Chas. M. Schwab, William H. Singer, Jos. D. Weeks, John F. Wilcox, Jos. R. Woodwell.

Hotel Headquarters.—The Monongahela House, in which, also, all sessions were held.

The opening session was held on Tuesday evening, February 18th, beginning at 8 o'clock. President Jos. D. Weeks, being also the chairman of the Local Executive Committee, welcomed the Institute to Pittsburgh in a few appropriate introductory words, and then proceeded to deliver the Presidential Address on the Invention of the Bessemer Process, Especially with Regard to the Relation to that Invention of William Kelly.

After some interesting discussion of this address, the President exhibited for inspection a sample of metallic manganese, containing 96 per cent. of that metal, and free from carbon, made by the process of Wahl and Greene, described by Mr. Garrison in a former paper before the Institute (*Trans.*, xxi., 887).

The President announced the appointment, as Scrutineers, to examine the ballots received for officers, and to report the result, Messrs. Thomas Robins, Jr., W. C. Brown and H. A. J. Wilkens.

After the election of new members (see list below), and various announcements by the Secretary and the Local Committee, the session was adjourned.

The second session was held on Wednesday evening, February 19th, when the following papers were read and discussed:

Notes on the Walrand-Legénisel Steel-Casting Process, by H. L. Hollis, Chicago, Ill.

The Embreville Estate of Tennessee, by Guy R. Johnson, Embreville, Tenn.

Notes on Conveying-Belts and Their Uses, by Thomas Robins, Jr., New York City.

The Magnetic Separation of Non-Magnetic Material, by H. A. J. Wilkens and H. B. C. Nitze, South Bethlehem, Pa.

The third session was held on Thursday morning, February 20th, when the following papers were presented in print:

The Accumulation of Amalgam on Copper Plates, by R. T. Bayliss, Marysville, Mont.

The Ore-Deposits of the Australian Broken Hill Consols Mine, Broken Hill, New South Wales, by George Smith, Broken Hill, New South Wales.

Note on Carbon-Bricks in the Blast-Furnace, by R. W. Raymond, New York City.

Notes on the Handling of Slags and Mattes at Smelting-Works in the Western United States, by William Braden, Helena, Mont.

The Cycle of the Plunger-Jig, by R. H. Richards, Boston, Mass.

The Effect of Vibration upon the Structure of Wrought-Iron (Continued Discussion).

The Effect of Additions of Titaniferous to Phosphoric Iron-Ores in the Blast-Furnace, by Auguste J. Rossi, New York City.

The Hydraulic Elevator at the Chestatee Mines, Georgia, by W. R. Crandall, Dahlonega, Ga.

The Assay by Prospectors of Auriferous Ores and Gravels by Means of Amalgamation and the Blow-Pipe, by W. H. Merritt, Toronto, Canada.

The discussion on the physics of cast-iron was continued by written and oral contributions from C. R. Baird & Co., Edward K. Landis, Alex. E. Outerbridge, Jr., Wm. R. Webster, Asa W. Whitney and D. Townsend, of Philadelphia, Pa.; Prof. R. C. Carpenter, Ithaca, N. Y.; Thomas D. West, Sharpsville, Pa.; William Kent, Passaic, N. J.; Guy R. Johnson, Embreville, Tenn.; Leonard Waldo, Bridgeport, Conn.; and Geo. S. Morison, Chicago, Ill.

The fourth and final session was held on Thursday afternoon, when the discussion of the physics of cast-iron was continued (some of the participants named above having spoken at this session), and the following papers were read by title:

Vein-Walls, by T. A. Rickard, Denver, Colo.

The Volatilization of Silver in Chloridizing-Roasting, by L. D. Godshall, Everett, Wash.

A Mechanical Drawer for Bee-Hive Coke-Ovens, by Robert A. Cook, New Brunswick, N. J.

The Newton-Chambers System of Saving the By-Products of Coke-Manufacture in Bee-Hive Ovens, by Robert A. Cook, New Brunswick, N. J.

Copper-Ores in the Permian Formation of Texas, by E. J. Schmitz, New York City.

Coal-Dust an Explosive Agent, by Donald M. D. Stuart, Bristol, England.

The Scrutineers reported the following persons to have been elected as officers of the Institute:

PRESIDENT.

E. G. SPILSBURY, ... Trenton, N. J.

VICE-PRESIDENTS.

(To serve two years.)

H. S. CHAMBERLAIN, ...	Chattanooga, Tenn.
ANTON EILERS, ...	Pueblo, Colo.
CHARLES KIRCHHOFF, ...	New York City.

MANAGERS.

(To serve three years.)

JAMES GAYLEY,	Pittsburgh, Pa.
JAMES F. KEMP,	New York City.
BENJ. SMITH LYMAN,	Philadelphia, Pa.

TREASURER.

THEODORE D. RAND,	Philadelphia, Pa.

SECRETARY.

ROSSITER W. RAYMOND,	New York City.

The Annual Report of the Council was presented as follows:

ANNUAL REPORT OF THE COUNCIL.

In accordance with the rules, the Council makes the following report to the Institute:

The financial statement of the Secretary and Treasurer shows receipts from all sources for the year ending February 1st (including $1201.77 on hand at the beginning of the year) of $27,616.29, and expenditures of $22,439.67, leaving a surplus of $5176.62, being a gain in the surplus of February 1, 1895, of $3974.85. In addition to this, the Treasurer holds U. S. bonds of the par value of $2900 and a special deposit of $4596, proceeds of U. S. bonds called in and paid by the government, which fund has not been permanently reinvested. The detailed statement of receipts and expenditures is as follows:

Receipts.

Balance from statement, February 1, 1895,		$1,201 77
Annual dues,	$21,360 26	
Life membership,	1,167 33	
Binding of *Transactions*,	1,785 10	
Sale of volumes of *Transactions*,	1,152 55	
" pamphlets,	683 34	
Electrotypes,	9 45	
Interest on U. S. bonds and deposits,	253 94	
Miscellaneous,	2 50	
		26,414 52
		$27,616 29

Disbursements.

Printing Volume XXIV. *Transactions*,	$2,967 51	
" pamphlet edition of papers,	2,638 80	
" mailing list,	24 00	
" circulars and ballots,	275 07	
Binding Volume XXIV. and miscellaneous volumes of *Transactions*,	1,860 60	
" exchanges,	47 81	
Engraving and electrotyping,	612 76	
Postage, including P. O. box-rent,	675 93	
Stationery,	169 02	
Rent,	800 00	
Express and freight charges,	1,125 82	
Telephone,	113 20	
Telegrams, cablegrams and car fare,	20 38	
Coal, ice and porters,	188 60	
Salaries, including clerks, stenographers and expenses of editing, and proof reading,	9,994 00	
Storage of *Transactions*,	111 10	
Special stenographers and expenses of meetings,	505 07	
Gas,	5 87	
Office supplies and repairs,	172 93	
Expenses mailing batches, etc.,	38 30	
Insurance,	59 40	
Refunding over-payments,	15 00	
Library additions,	18 50	
		$22,439 67
Balance,		5,176 62
		$27,616 29

Two meetings were held during the year; the Annual meeting beginning at New York in February and continued by adjournment to Florida in March and April, and the Atlanta meeting of October. The proceedings and papers of these meetings, already published and distributed to members, so clearly show their delightful and interesting character as to require no further comment on the part of the Council.

Changes in membership have taken place during the year as follows: One honorary member, 142 members and 18 associates have been elected; 3 associates have become members; the deaths of 2 honorary members, 19 members and 1 associate have been reported; 60 members and 5 associates have resigned; and 94 members and 5 associates have been dropped for continued default in the payment of dues. These changes are tabulated as follows, showing a net loss of 25 in total mem-

bership, which is much more than made up by the large number of applications already approved for election at the Annual meeting now pending:

	H. M.	F. M.	M.	A.	Totals.
At date of last report...	14	38	2216	169	2437
Gains: By election................	1	......	142	18	161
change of status.....	......	......	3	......	3
Losses: By resignation...........	......	......	60	5	65
dropping...............	......	......	94	5	99
change of status.....	......	......	......	3	3
death..................	2	......	19	1	22
Total gains.........................	1	......	145	18	164
Total losses	2	......	173	14	189
Present membership..............	13	38	2188	173	2412

The list of deaths comprises the names of. Antonio del Castillo and Franz Posepny, honorary members, and the following members and associates: Arthur Chanute (1882), Ellis Clark (1874), C. M. Conyngham (1891), Eckley B. Coxe (1871), James G. Dagron (1884), A. H. DeCamp (1883), E. G. DeCrano (1872), F. J. Dominick (1882), F. P. Gracey, (1890), John Heard, Jr. (1883), M. B. Jamieson (1895), Edward Jones (1887), R. H. Lee (1876), J. M. Reid (1889), A. W. Sims (1885), John H. Sprow (1893), Andrew Stevens (1894), D. M. Thomas (1892), J. Fraser Torrance (1876), and H. Walker (1894).

This list includes names of high professional and scientific distinction. The character and work of Prof. Posepny, and of the lamented Eckley B. Coxe, one of the founders of the Institute, have been appropriately recognized in biographical notices in the *Transactions*. Messrs. Clark, Lee and Torrance ranked among our veteran members, having been connected with the Institute for 20 years or more. Mr. Clark had contributed to the *Transactions* the following valuable papers: "Shaft-Surveying in the Brown Hematite Mines of Northampton County, Pa." (*Trans.*, vii., 139); "The Great Blast at Glendon" (*Trans.*, vii., 266); "Ore-Dressing and Smelting at Przibram, Bohemia" (*Trans.*, ix., 420); "Notes on the Progress of Mining in China" (*Trans.*, xix., 571); "The Silver-Mines of Lake Valley, New Mexico" (*Trans.*, xxiv., 138).

The amendments to the rules of which notice had been

given in writing at the Atlanta meeting, were discussed, and adopted (after amendment) in the form following, to be submitted by mail to the members and associates of the Institute for final adoption.

RULE II.—It is proposed to amend this rule by adding the following: "and members or associates whose post-office addresses are outside of the United States, Canada and Mexico shall not be entitled to vote by mail except upon proposed amendments to the Rules."

RULE III.—It is proposed to amend this rule by striking out the words "at the annual meeting," and substituting the words "on the first day of each calendar year."

RULE VI.—It is proposed to amend this rule by striking out the provision for special meetings, and changing the second sentence of the rule so that it shall read, "other meetings shall be held at such times and places as the Council shall select, and notice of all meetings shall be given," etc.

RULE VII.—It is proposed to amend this rule by making the title "Papers and Publications," and by inserting after the first paragraph the following: "The published papers and volumes of Transactions shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the rules."

Also, by adding to the rule the following: "Nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion, or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute as provided in these rules; *Provided, however*, that committees may be appointed by the Council or the Institute to make investigations, and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports."

After the adoption of a resolution directing the Secretary to acknowledge by letter the courtesies received by the Institute, the meeting was adjourned.

Members and Associates Elected.

The following persons were made members or associates by election during the sessions of the meeting:

Members.

R. Wilson Anderson,	Pittsburgh, Pa.
Everett D. Arnold,	Troy, N. Y.
H. B. Barnhart,	Homestead, Pa.
Joseph Barrell,	South Bethlehem, Pa.
J. Basadre,	Lima, Peru.
R B. Beahm,	Bethlehem, Pa.
Henry Lynch Bellam,	Anaconda, Mont.
George R. Bentley,	Harrisburg, Pa.
A. Newton Bluett,	Lökling, Norway.
Ernest Elmer Breisch,	Winton, Pa.
Chauncey E. Butler,	Socorro, New Mexico.
William Curtis Butler,	Everett, Washington.
Benson M. Caldwell,	Bridgeport, Ohio.
Charles Merritt Case.	Minneapolis, Minn.
John Allen Cass,	Schenectady, N. Y.
Walton Clark,	Philadelphia, Pa.
H. M. Cole,	East Helena, Mont.
Henry Francis Collins,	Bathurst, New South Wales.
J. Parker Corbus,	Douglas Island, Alaska.
Walter Currie,	Buluwayo, So. Africa.
A. L. Dean,	East Helena, Mont.
Isaac W. Frank,	Pittsburgh, Pa.
George Blanchard Fry,	London, England.
Charles G. Griffith,	Helena, Mont.
J. McC. Henderson,	London, England.
William Alfred Heyward,	Anaconda, Mont.
B. C. Hinman,	Boulder, Colo.
Brian Hooker,	Hannan's, W. Australia.
Azor R. Hunt,	Munhall, Pa.
Frederick Irwin,	Silver City, Idaho.
Charles Borrows Jacobs,	Schenectady, N. Y.
G. B. Jacobs,	Charcas, Mexico.
John Jarrett,	Pittsburgh, Pa.
John Jobson,	Port Perie, So. Australia.
John Kennedy,	Hobart, Tasmania.
George Labram,	Kimberly, So. Africa.
S. C. Leonard,	Niles, Ohio.
Robert H. Lyman,	Nueva Salamanca, Santiago de Cuba.
John McConnell,	Swissvale, Pa.
Alexander George McKenna,	Demmler, Pa.
Louis McKenzie,	Wheeling, W. Va.
D. W. McNaugher,	Pittsburgh, Pa.
Angus R. Mackay,	Deadwood, So. Dakota.
George Wickham Metcalfe,	Embreville, Tenn.
Theo. Morrison,	Braddock, Pa.
Charles Murray,	Johannesburg, So. Africa.
Frederick S. Pearson,	New York City.
Harvey Pridham,	Coolgardie, W. Australia.

B. S. Revett, ...	Breckenridge, Colo.
Oliver G. Ricketson, . . .	Pittsburgh, Pa.
W. D. Sargent, ...	Chicago, Ill.
Herman Henry Schlapp, . .	Melbourne, Australia.
Carl Scholz, ...	Mammoth, W. Va.
F. H. Slocum, ...	Pittsburgh, Pa.
Alfred Smedley, ...	Oil City, Pa.
J. Ramsay Sperr, ...	Pittsburgh, Pa.
Alexander Stewart, . . .	Broken Hill, New South Wales.
Howard Noble Stockett, . .	Clarksburg, W. Va.
James F. Tarwater, . . .	Rookwood, Tenn.
John W. Taylor, ...	Tin Cup, Colo.
William Stephens Thomas, . .	Belt, Mont.
Joseph E. Thropp, Jr., . . .	Everett, Pa.
Edward N. Trump, . . .	Syracuse, N. Y.
Charles C. Upham, . . .	New York City.
Samuel M. Vauclain, . . .	Philadelphia, Pa.
Seymour Waterhouse, . . .	San Francisco, Cal.
Charles C. Weir, ...	New York City.
J. Price Wetherill, . . .	South Bethlehem, Pa.
H. M. Whitney, ...	Boston, Mass.
Albert D. Wilkins, . . .	Pittsburgh, Pa.
Theodore Kirkland Wilkinson, .	Anaconda, Mont.
John A. Wood, Jr.. . . .	Pittsburgh, Pa.
Lee S. Wood, ...	Rico, Colo.
Samuel Woods, ...	Pittsburgh, Pa.
Adolph A. Zimmerman, . .	Elkhorn, Mont.

Associates.

John T. Callaghan, Jr., . . .	Washington, D. C.
William Stewart Davison, . .	Brooklyn, N. Y.
John Lindsay Ferguson, . .	Philadelphia, Pa.
Frank Humel, ...	Cleveland, Ohio.
Robert Henry Jeffrey, . . .	London, England.
John Lilly, ...	Lambertville, N. J.
William Thaxter Thompson, .	Fairview, British Columbia.
William D. Thornton, . . .	Butte, Mont.
John Schofield Wallace, . .	Bethlehem, Pa.
Carlos Yglesias y Castro, . .	Bethlehem, Pa.

Associates made Members.

F. F. Amsden, ...	South St. Louis, Mo.
Bertrand S. Summers, . . .	Chicago, Ill.
Robert S. Weir, ...	Hermosillo, Mexico.

Excursions and Entertainments.

Wednesday, February 20th, was spent in visiting the Homestead Steel-Works, the Westinghouse Electric Works, and the Edgar Thomson Steel-Works. The party left Pittsburgh by special train, furnished by the Pennsylvania Railroad, at 9 A.M., and arrived in Pittsburgh at 5 P.M.

On Thursday afternoon, the ladies were entertained with carriage-drives, and visits to the Carnegie Library, the Phipps Conservatory and the Parks.

On Thursday evening a subscription banquet was held at the Monongahela House. The attendance was large, and the goodly proportion of ladies present added grace and spirit to the occasion.

On Friday, the party was conveyed by special train, leaving Pittsburgh at 9 A.M., to the Glass Works of McKee Brothers, at Jeannette, and thence to the Latrobe Steel-Works, Latrobe (stopping en route to inspect the Newton-Chambers coke-ovens of that company). After luncheon at the Latrobe works, the train proceeded to the works of the Pennsylvania Salt Company, at Natrona, and returned thence to Pittsburgh, arriving at 5 P.M.

Cordial invitations were received from the following establishments, of which many members, singly or in small parties, availed themselves:

Youghiogheny Coal Company, United States Glass Company, Forest Oil Company and Gas Wells, Standard Coke Works of the H. C. Frick Coke Company, Otto-Hoffman Coke-Ovens, A. Garrison Foundry and Machine Works, Monongahela Furnaces and Steel-Works, Bridgewater Gas Company, McIntosh, Hemphill & Company's Foundry and Machine Works, Star Tin Plate Works, Jones and Laughlins Steel-Works, Oliver & Roberts Wire Company.

Members, Associates and Guests Registered.

The following persons were registered at hotel headquarters:

Taylor Allderdice.
R. Wilson Anderson.
E. D. Arnold.
F. E. Bachman.
Charles Bailey.
David Baker.
P. T. Berg.
George Best.
W. H. Bradley.
William Clinton Brown.
J. M. Camp.
Henry R. Cassel.
George H. Clapp.
E. T. Clymer.
Lee S. Clymer.
W. B. Cogswell.
James M. Colwell.
Charles Connor.
P. H. Conradson.
Robert A. Cook.
W. E. Corey.
Eugene Coste.
Justice Cox, Jr.
Fred. Crabtree.
Walter Crafts.
F. Hearne Crockard.
Ralph Crooker, Jr.
Benedict Crowell.

Benjamin Crowther.
S. W. Croxton.
James Darsie.
David T. Day.
George A. Dean.
C. E. Dewey.
F. A. Emmerton.
W. E. C. Eustis.
Richard Eyre.
George Faunce.
E. M. Ferguson.
William Forsyth.
Lewis W. Francis.
Willard Fuller.
H. D. Gamble.
James Gayley.
P. H. Griffin.
William Hainsworth.
Charles M. Hall.
George W. Harris.
Edward R. Hewitt.
Henry D. Hibbard.
H. L. Hollis.
J. F. Holloway.
Howard M. Hooker.
Theo. M. Hopke.
W. S. Hungerford.
Alfred E. Hunt.
Robert W. Hunt.
Harold Jeans.
G. R. Johnson.
J. Edward Keeler.
Fred. C. Keighley.
J. G. Kelly.
Hugh Kennedy.
Julian Kennedy.
William Kent.
D. G. Kerr.
Charles Kirchhoff.
E. L. Kurtz.
H. M. Kurtz.
H. M. Lash.
R. G. Leckie.
George M. Lehman.
N. Lilienberg.
Stuart Lindsley.
E. V. McCandless.
H. McCormick, Jr.
Vance C. McCormick.
Charles McCrery.
J. C. McDowell.
E. L. McGary.
J. King McLanahan.
James MacNaughton.
William Metcalf.
G. S. Morison.
C. A. Morrison.
M. G. Morse.
C. B Murray.
J. W. Murray.
William G. Neilson.
W. H. Nimick.
H. B. C. Nitze.
H. M. Norris.
G. S. Page.
E. W. Parker.
F. S. Pearson.
William H. Peirce.
Samuel Peters.
Francis C. Phillips.
J. J. Pierce.
J. C. Porter.
W. C Ralston.
Erskine Ramsay.
Theodore D. Rand.
R. W. Raymond.
W. H. Rea.
Arnold K. Reese.
F. B. Richards.
Robert H. Richards.
Percival Robertson.
Thomas Robins, Jr.
Cyrus Robinson.
D. S. Robinson.
T. W. Robinson.
Luther A. Roby.
Auguste J. Rossi.
William Rothoff.
T. S. Russell.
O. P. Scaife.
W. L. Scaife.
F. Z. Schellenberg.
James Scott.
S. Taylor Sheaffer.
P. G. Shook.
Clement G. Smith.
J. W. Smith.
Pemberton Smith.
T. Guilford Smith.
E. G. Spilsbury.
C. E. Stafford.
H. R. Stanford.
A. A. Stevenson.
John Stevenson, Jr.
W. R. Stirling.
Edward G. Stoiber.
J. Sulton.
Benjamin Talbot.
George E. Thackray.
J. M. Thomas.
John L. Thompson.
Robert M. Thompson.

Joseph E. Thropp, Jr.
Theodore Tonnelé.
D. Townsend.
Edward N. Trump.
W. R. Tucker.
James W. Tyson, Jr.
Edward A. Uehling.
J. S. Unger.
Leonard Waldo.
Robert L. Walker.
T. B. Walker.
William R. Webster.
Joseph D. Weeks.
Charles H. Wellman.
S. T. Wellman.
John Price Wetherill.
Charles Wharton, Jr.
Henry M. Whitney.
John F. Wilcox.
William H. Wiley.
Frank Williams.
H. A. J. Wilkens.
John Wilkes.
A. D. Wilkins.
William Wilkins.
W. G. Wilkins.
Jones Wister.
James P. Witherow.
E. T. Wood.
J. A. Wood, Jr.
Samuel Woods.

Proceedings of the Seventy-First Meeting, Colorado, September, 1896.

General Executive Committee.

Richard Pearce, *Chairman;* H. Van F. Furman, J. A. Porter, T. A. Rickard, B. B. Lawrence, H. V. Pearce, T. B. Stearns, P. H. Van Diest.

Local Committees.

Denver.—J. B. Grant, *Chairman;* W. L. Austin, F. Dillingham, H. W. Hobson, N. P. Hill, H. R. Wolcott, D. Sheedy, J. F. Vaile, G. C. Munson, H. A. Vezin, F. E. Hartman, A. Gove, J. W. Nesmith, D. H. Moffat, S. T. Smith, H. H. Lee, C. J. Hughes, Jr., H. E. Wood, W. S. Ward, T. E. Schwarz, E. E. Burlingame, John Campion, J. B. Farish, J. D. Hawkins, G. E. Randolph, S. Guggenheim, S. F. Rathvon, G. W. Pierce, E. O. Wolcott, C. H. Toll, W. B. Page, F. E. Shepard, E. T. Jeffery, J. A. Thatcher, J. A. Kebler, T. M. Patterson, T. S. McMurray, Eben Smith, A. G. Gorham, E. L. Foucar.

Cripple Creek.—C. J. Moore, *Chairman;* F. T. Freeland, Wm. Weston, J. S. Luckcraft, William Bainbridge, F. W. Howbert, J. M. Parker, C. Keith, E. N. Hawkins, O. B. Willcox, M. L. Chapman, E. Skewes, T. R. Countryman, H. Steele, N. H. Cone, B. Hunt.

Victor.—E. L. White, J. B. Cunningham, W. B. Briggs, A. B. Bumstead, J. Doyle, J. Harnan, J. H. Mackenzie, T. R. Woodbridge.

Colorado Springs.—R. J. Bolles, *Chairman;* W. Devereux, J. H. Bolles, William Strieby, C. M. MacNeill, T. Parrish, A. L. Dickerman, C. E. Palmer, W. F. Fisher, J. W. Proudfit, J. J. Hagerman, W. B. Donaldson.

Florence.—P. Argall, S. F. Rathvon, G. Robinson, C. F. Lacombe.

Pueblo.—A. S. Dwight, *Chairman;* Anton Eilers, W. W. Allen, T. W. Robinson, C. S. Robinson, Karl Eilers, F. E. Harnden, E. P. Mathewson, W. H. Howard, P. A. Mosman, E. W. Johnson.

Central City.—John Best, E. Le N. Foster, J. W. Bostwick, Forbes Rickard, A. L. Collins, Fred. Kruse.

Golden.—Regis Chauvenet, H. Van F. Furman.

Aspen.—D. W. Brunton, Francis T. Freeland, E. M. Rogers, Frank Bulkley, S. I. Hallett, F. M. Taylor, E. Dunbar Wright, C. E. Doolittle, Geo. S. Newman.

Hotel Headquarters in Denver.—The Brown Palace Hotel.

Hotel Headquarters at Cripple Creek.—The National Hotel.

The first session was held at the Brown Palace Hotel, Denver, on Monday, September 21st, at 2 P.M.

Mr. Richard Pearce, Chairman of the Local Executive Committee, welcomed the Institute to Colorado, apologizing for the absence of the Governor of the State, who had promised to perform this office, but was prevented by arduous labors connected

with the strike at Leadville, and the critical condition of affairs at that place involving executive vigilance and action.

After a suitable reply by the Secretary, Mr. Pearce, as an ex-President of the Institute, assumed the chair in the absence of the President.

The following announcements were made from the Council:

An offer on the part of the Colorado Scientific Society of the free use of its rooms in the Boston Building by visiting members and guests.

A similar offer on the part of the Denver Society of Civil Engineers.

Announcement that the next meeting of the Institute would be held in Chicago in February, 1897.

Announcement of the death of Prof. Auguste Daubrée, an honorary member of the Institute, and of a biographical notice, to be printed hereafter.

Announcement of the death of Vice-President Charles A. Stetefeldt, and presentation of a biographical notice by the Secretary.

Announcement of the death of ex-Vice-President J. F. Holloway, and of a biographical notice by J. F. Lewis, Chicago, Ill.

The following papers were read and discussed:

The Development of Colorado's Mining Industry, by T. A. Rickard, State Geologist, Denver, Colo.

The Occurrence and Behavior of Tellurium in Gold-Ores, more Particularly with Reference to the Potsdam Ores of the Black Hills, South Dakota, by Frank C. Smith, Rapid City, South Dakota. (Presented by the Secretary in the absence of the author.)

Electrical Mining in the Rocky Mountain Region, by Irving Hale, Denver, Colo.

Additions to the Power-Plant of the Standard Consolidated Mining Company, by Robert G. Brown, Bodie, Cal.

The discussion of these two papers was adjourned to the next session.

After adjournment, Mr. H. B. C. Nitze, of South Bethlehem, Pa., exhibited in operation a working-model of the Wetherill magnetic separator, described in the paper of Messrs. Wilkens and Nitze, presented at the Pittsburgh meeting, on "The Magnetic Separation of Non-Magnetic Material."

The second session was held at the Brown Palace Hotel on Tuesday evening, September 22d; Vice-President H. S. Chamberlain occupied the chair at this and subsequent sessions. After an exceedingly interesting discussion of the papers of Mr. Hale and Mr. Brown, presented at the preceding session, the following paper was presented by the Secretary in the absence of the author:

Silver-Losses in Cupellation, by L. D. Godshall, Everett, Washington.

In connection with the same subject, the following paper was read and discussed:

An Improved Assay-Muffle, by Arthur S. Dwight, Pueblo, Colo.

The following paper was read and discussed:

Sketch of a Portion of the Gunnison Gold-Belt, Including the Vulcan and Mammoth Chimney Mines, by Arthur Lakes, Denver, Colo.

The following papers were presented in printed form:

The Smuggler Union Mines, Telluride, Colorado, by J. A. Porter, Denver, Colo.

Gold in Granite and Plutonic Rocks, by W. P. Blake, Tucson, Ariz.

The Enterprise Mine, by T. A. Rickard, Denver, Colo.

Further Notes on the Alabama and Georgia Gold-Fields, by W. M. Brewer, Atlanta, Ga.

Gold in the Guyanas, by Henry G. Granger, Buenaventura, Colombia, S. A.

Note on a Shaft-Fire and Its Lesson, by Robert G. Brown, Bodie, Cal.

Action of Blast-Furnace Gases Upon Various Iron-Ores, by O. O. Laudig, Buffalo, N. Y.

The Microstructure of Steel and the Current Theories of Hardening, by Albert Sauveur, Chicago, Ill.

The Actual Accuracy of Chemical Analysis, by F. P. Dewey, Washington, D. C.

Rapid Section-Work in Horizontal Rocks, by M. H. Campbell, Washington, D. C.

Laboratory-Note on the Heat-Conductivity, Expansion and Fusibility of Fire-Brick, by J. D. Pennock, Syracuse, N. Y.

The Bertrand-Thiel Open-Hearth Process, by Joseph Hartshorne, Philadelphia, Pa.

Note on Copper in Iron and Steel, by R. W. Raymond, New York City.

Traces of Organic Remains from the Huronian (?) Series at Iron Mountain, Michigan, etc., by W. S. Gresley, Erie, Pa.

Also communications in discussion of the following papers:

Mr. Laudig's paper (above mentioned).

Mr. Bayliss's paper (Pittsburgh meeting) on the Accumulation of Amalgam on Copper Plates.

Mr. Thackray's paper (Atlanta meeting) on the Determination of Phosphorus in Steel.

Mr. Schmitz's paper (Pittsburgh meeting) on Copper-Ores in the Permian of Texas.

Mr. Rickard's paper (Pittsburgh meeting) on Vein-Walls.

And the following papers were read by title:

Occurrence and Treatment of Gold-Ores in Park County, Colorado, by B. Sadtler, Denver, Colo.

Occurrence of Gold-Ores in the Rainy River District, Ontario, Canada, by William Hamilton Merritt, Toronto, Can.

The third session was held in the National Hotel at Cripple Creek, on Wednesday evening, September 23d. The following papers were read and discussed:

Faulting and Accompanying Features Observed in Glacial Gravel and Sand in Southern Michigan, by Carl Henrich, Noble, Ill.

The following papers were presented in printed form:

A Modern Silver-Lead Smelting-Plant, by L. S. Austin, Denver, Colo.

The Sulphuric-Acid Process of Treating Lixiviation-Sulphides, by F. P. Dewey, Washington, D. C.

And the following were read by title:

Ore-Shoots of Cripple Creek, Colorado, by Edward Skewes, Cripple Creek, Colo.

Phosphate-Deposits of Arkansas, by John C. Branner, Stanford University, Cal.

The Use of the Tremain Steam-Stamp in Amalgamation, by Edwin A. Sperry, Gunnison, Colo.

Magnetic Observations in Geological Mapping, by H. L. Smyth, Cambridge, Mass.

Some Mines of Rosita and Silver Cliff, Colorado, by S. F. Emmons, Washington, D. C.

The fourth and final session was held at the School of Mines, Golden, Colo., on Monday evening, September 28th.

The following paper was read and discussed:

Solution and Precipitation of Cyanide of Gold, by S. B. Christy, Berkeley, Cal.

The following papers were presented in pamphlet form:

Laboratory-Tests in Connection with the Extraction of Gold from Ores by the Cyanide-Process, by Howard Van F. Furman, Denver, Colo.

The Cyanide-Process in the United States, by George A. Packard, Boston, Mass.

The Concentration of Ores in the Butte District, Montana, by Charles W. Goodale, Butte, Mont.

Middle-Product Jig, by E. G. Tuttle, Newark, N. J.

Excentric Jig, with Adjustable and Automatic Lower Discharge Arranged for the Full Width of the Bed and for One or More Compartments, by E. G. Tuttle, Newark, N. J.

After the adoption of a resolution instructing the Secretary to express to the several individuals and corporations concerned the thanks of the Institute for courtesies received, the meeting was adjourned.

Members and Associates Elected.

The following persons were elected by mail, April, 1896.

Members.

Frank Anderson,	Salt Lake City, Utah.
Richard W. Bailey,	Pottstown, Pa.
Lafayette Grant Burton,	Salt Lake City, Utah.
James Macdonald Calderwood,	Helena, Mont.
E. H. Cook,	Globe, Ariz.
Konrad E. R. Engel,	Essen, Germany.
James Hart Fawcett,	Coolgardie, W. Australia.
Philip L. Foster,	New York City.
Edward Halse,	Coolgardie, W. Australia.
Herbert C. Hoover,	San Francisco, Cal.
Bertram Hunt,	Denver, Colo.
Joseph MacDonald,	Gem, Idaho.
J. Lawrence Malm,	Cleveland, Ohio.
Burdett Moody,	Lead, S. Dakota.
Thomas D. Murphy,	Harqua Hala, Ariz.
Chester Wells Purington,	Washington, D. C.
Edward Danforth Self,	South Orange, N. J.
William H. Tonking,	Port Oram, N. J.
Charles Van Isschot,	Guayaquil, Ecuador, S. A.

Associates.

George F. C. Hosking,	Whangarei, Auckland, N. Z.
J. W. Stearns,	Denver, Colo.

The following persons were elected as members or associates at the sessions of the meeting.

Members.

Truman H. Aldrich, Jr.,	Birmingham, Ala.
Miltiades Th. Armas,	Sonora, Mexico.
Richard M. Atwater, Jr.,	London, England.
Ebenezer C. Babbitt,	Cripple Creek, Colo.
Franklin Baker, Jr.,	South Chicago, Ill.
Leon H. Barnett,	Rico, Colo.
James Brady,	Victoria, British Columbia.
Maurice Andrews Bucke,	Kaslo, British Columbia.
William Carkeek,	Butte, Mont.
William A. Carlyle,	Victoria, British Columbia.
Eduardo Justo Chibas,	New York City.
Hopewell Clarke,	St. Paul, Minn.
T. Launcelot Dawson,	Panama, S. A.
Samuel Dixon,	Macdonald, W. Va.
John R. Don,	Dunedin, New Zealand.
Charles Horace Doolittle,	Silverton, Colo.
Fernand J. Dorion,	Sonora, Mexico.
John Essen,	Chatteris, England.
Arthur Wiley Evans,	Petros, Tenn.
Wallace Fairbank,	Congress, Ariz.
James Hart Fawcett,	Coolgardie, W. Australia.
C. N. Fenner,	New York City.
Robert Jay Forsythe,	Cambridge, Mass.
Wilson P. Foss,	Haverstraw, N. Y.
John D. Gilchrist,	Iron Mountain, Minn.
Eugene Louis Giroux,	Jerome, Ariz.
Meade Goodloe,	Congress, Ariz.
Gustave M. Gouyard,	Denver, Colo.
John A. Grant,	Lost Creek, Pa.
Julius Grillo,	Hamborn, Germany.
Edgar Hall,	Queensland, Australia.
Rasmus Hanson,	Eureka, Colo.
C. Leland Harrison,	Philadelphia, Pa.
Victor C. Heikes,	Sunshine, Utah.
Martin Joseph Heller,	San Francisco, Cal.
Victor G. Hills,	Cripple Creek, Colo.
William J. Isaacson,	Cincinnati, Ohio.
Charlie Eakin James,	Chattanooga, Tenn.
L. J. W. Jones,	Denver, Colo.
Thomas S. Jones,	Salt Lake City, Utah.
Carl Koelle,	Argentine, Kan.
Arthur Lakes,	Denver, Colo.

Allan G. Lamson,	Salt Lake City, Utah.
James G. Lawn,	Johannesburg, S. African Rep.
Henry P. Lowe,	Denver, Colo.
Nathan T. Mansfield,	Telluride, Colo.
Emile M. C. du Marais,	Rhone, France.
Thomas Bruce Marriott,	Brazil, S. A.
William B. Milliken,	Cripple Creek, Colo.
H. W. Mitchell,	Pueblo, Colo.
Harry Sanderson Mulliken,	Pilot Bay, British Columbia.
Charles H. Munger,	Ironwood, Mich.
John W. Nesmith,	Denver, Colo.
Thomas Nichol,	Glenjean, W. Va.
Stanley H. Pearce,	Denver, Colo.
Edmund B. Preston,	San Francisco, Cal.
Dr. Charles Pret,	Durango, Colo.
R. Recknagel,	Bourne, Oregon.
George Samuel Rice,	Ottumwa, Iowa.
Edgar Rickard,	Keswick, Cal.
T. P. Rigney,	Cripple Creek, Colo.
John Ross, Jr.,	Sutter Creek, Cal.
B. Sadtler,	Denver, Colo.
Edwin F. Saxman,	Latrobe, Pa.
Frank Edward Shepard,	Denver, Colo.
S. Rodmond Smith,	Wilmington, Del.
Stephen Westropp Stacpoole,	Mexico, Mexico.
William A. Stanton,	Pittsburgh, Pa.
John Marius Timm,	Pilgrims' Rest, South Africa.
Luther Wagoner,	San Francisco, Cal.
William Watson,	Johannesburg, S. African Rep.
William Weston,	Cripple Creek, Colo.
Charles Livy Whittle,	Cambridge, Mass.
James Wilding, Jr.,	Chihuahua, Mexico.
Walter H. Wiley,	Idaho Springs, Colo.
Richard Henry Williams,	Guanajuato, Mexico.
William Duncan Williamson,	Greenock, Scotland.

Associates.

Alfred Chester Beatty,	New York City.
Robert Montague Browne,	Perth, W. Australia.
Charles E. Finney,	Argentine, Kan.
William D. Gordon,	Chicago, Ill.
John Gross,	Elyria, Colo.
Frank H. Lerchen,	Denver, Colo.
Frank W. Popple,	Denver, Colo.
Edward Coppée Thurston,	Freiberg, Saxony.
Gustave Thorkildsen,	Chicago, Ill.

Associates Made Members.

Richard S. McCaffery,	New York City.
Russell T. Mason,	Houghton, Mich.
Joseph Philips, Jr.,	San José, C. A.
Carlos Yglesias y Castro,	San José, C. A.

Excursions and Entertainments.

Monday evening, September 21st, the members and guests of the Institute were gracefully entertained at an afternoon tea at the residence of Mr. Richard Pearce.

Tuesday morning, September 22d, an excursion was made by special train to the works of the Boston and Colorado Smelting Company, and the Omaha and Grant Smelting and Refining Company.

Tuesday afternoon, an excursion was made by special train to Fort Logan, where the party was enabled, through the courtesy of Gen. Wheaton, commanding the department, and Col. Merriam, commanding the post, to witness not only a dress-parade of the cavalry and infantry there stationed, but also a most interesting exhibition of the conditions and operations of actual field-service, including the making and "striking" of camp by the infantry, the handling of horses by the cavalry, bareback-riding, etc.

Wednesday, September 23d, a special train, leaving Denver at 9 A.M., conveyed the Institute party, *via* Colorado Springs, to Cripple Creek, which was reached late in the afternoon. The remainder of the day, the whole of Thursday, and the morning of Friday, were spent in visiting the leading mines and reduction-works of Cripple Creek and Victor, including the Anchoria-Leland, Rebecca, Portland, Independence, Isabella, Strong, Anaconda, Moose, Elkton and other mines, and the Taylor and Brunton and Victor sampling-works, the Brodie cyanide-works, and the El Paso chlorination-works (at Gillett).

Thursday evening, September 24th, a banquet was given by the Cripple Creek Local Committee to the visiting members and guests of the Institute at the National Hotel. In view of the fact that the town had been but recently almost destroyed by conflagration, and that the very building in which this banquet took place was not yet finished for regular occupancy, the complete success of the entertainment reflected special credit on the energy and skill of the Local Committee.

Friday evening, September 25th, an elegant reception was given to the visiting members and guests of the Institute by the Colorado Springs Local Committee, at the El Paso Club, Colorado Springs, after which, at 1 A.M., the special train proceeded to Pueblo for breakfast, and thence to Florence.

Saturday morning, September 26th, was spent in visiting the cyanide-works of the Metallic Extraction Company, at Cyanide station, and the oil-wells and refinery of the United Oil Company at Florence. At the latter place an interesting array of products was exhibited, and the visitors were adorned with nosegays of wild flowers by the children; and baskets of magnificent fruits, grown at Florence, were placed on the train before its departure.

Arriving again in Pueblo at 1.30 P.M., the visitors became the guests of the Pueblo Local Committee. The afternoon was spent in excursions to the steel-works of the Colorado Fuel and Iron Company and the lead-smelting works of the Colorado Smelting Company, the Pueblo Smelting and Refining Company, and the Philadelphia Smelting and Refining Company. At 6.30 P.M. a supper was given to the party in the beautiful Minnequa club-house, on the banks of Minnequa Lake, which was picturesquely illuminated with electric lights. The supper was followed by a reception and dancing, after which the party was conveyed in trolley-cars to the Union Depot, and the special train left for Colorado Springs and Denver shortly before midnight.

Sunday was spent at Manitou by many of the party, and a considerable number made the trip to the top of Pike's Peak.

Monday morning, September 28th, a special train left Denver for Black Hawk, where lunch was furnished, and a special train on the Gilpin tramway conveyed the party to visit the Gold Coin, Gregory-Bobtail, Saratoga and California mines, and to enjoy the grand prospect from the summit of the divide.

Monday afternoon the party returned to Golden, where an elegant supper and reception were given by the Local Committee at the School of Mines, the building, collections and appointments of which were inspected with admiration and pleasure. After a brief evening session, the party returned to Denver, arriving late at night.

Tuesday morning, September 29th, a considerable number of visiting members and guests left Denver, *via* the Colorado Midland route, for Glenwood Springs and Aspen. The luxury of the Hotel Colorado and of the famous baths and swimming pool was thoroughly appreciated.

Wednesday afternoon was spent at Aspen, where, by virtue

of the admirable arrangements of the Local Committee, the party was enabled to inspect the Cowenhoven tunnel and the mines connected with it, the electric-power plants which have made Aspen famous, and some of the concentration- and reduction-works. After enjoying, at 5.30 P.M., a dinner given by the Local Committee, the guests left for Glenwood at 7.05 P.M.

Thursday was spent in returning to Pueblo and Denver *via* the Rio Grande route, including the passage of the cañon of the Grand, the Arkansas valley and the Royal Gorge.

Members, Associates and Guests Registered.

The following list is based almost wholly on the register at the Hotel-Headquarters in Denver, and doubtless does not include the names of many who participated in the sessions at other places:

William S. Ackerman.
William A. Akers.
Walter W. Allen.
George A. Anderson.
Philip Argall.
J. R. Ashley.
William Bainbridge.
John Birkinbine.
Richard J. Bolles.
Robert G. Brown.
William Clinton Brown.
D. W. Brunton.
F. G. Bulkley.
E. E. Burlingame.
Donald W. Campbell.
George Cannon.
Henry R. Cassel.
Franz Cazin.
H. S. Chamberlain.
Regis Chauvenet.
J. Morgan Clark.
Arthur L. Collins.
N. H. Cone.
Edgar S. Cook.
William Coumerilh.
T. R. Countryman.
B. L. Creves.
Frank H. Crockard.
W. W. J. Croze.
N. H. Darton.
William C. Davis.
David T. Day.
W. S. De Camp.
Theodore Dengler.
W. B. Devereux.
C E. Dewey.
Alton L. Dickerman.
Arthur S. Dwight.
A. Eilers.
Karl Eilers.
W. E. C. Eustis.
Franz Fohr.
Ernest Le Neve Foster.
Edouard L. Foucar.
Francis T. Freeland.
George B. Fress.
Howard Van F. Furman.
Herman Garlichs.
Austin G. Gorham.
G. M. Gouyard.
James B. Grant.
H. M. Griffin.
Franklin Guiterman.
Irving Hale.
F. E. Harnden.
Frank E. Hartman.
Edwin N. Hawkins.
J. D. Hawkins.
G. C. Hewitt.
Rossiter Howard.
F. W. Howbert.
Benjamin P. Howell.
Bertram Hunt.
C. E. James.

R H Jeffrey.
Walter P. Jenney.
E. W. Johnson
Julian A. Kebler.
Edmund B. Kirby.
G. Lavagnino.
Benjamin B. Lawrence.
H. C. Lay.
I. P. Lihme.
Charles H. Livingston.
Richard W. Lodge.
Henry P. Lowe.
John S. Luckraft.
Edwin Ludlow.
William R. McIlvain.
William McKell.
D. McVichie.
E. du Marais.
E. P. Mathewson.
Charles C. Mattes.
De Courcy May.
Frank P. Mills.
Charles J. Moore.
B. F. Morley.
P. A. Mosman.
J. Stanley Muir.
George C. Munson.
H. B C. Nitze.
George S. Oliver.
Edward W. Parker.
Richard Pearce.
Edmund C. Pechin.
R. A. F. Penrose, Jr.
William H. Pettee.
William Plummer.
J. A. Porter.
George E. Potts.
R. W. Raymond.
George S. Raymer.
George S. Rice.
Forbes Rickard.
T. A. Rickard.
Heinrich Ries.
James D. Robertson.
C. Snelling Robinson.
E. M. Rogers.
John E. Rothwell.
T. E Schwarz.
H. J. Seaman.
A. W. Sheafer.
T. E. Shepard.
Edward Skewes.
George W Small.
Frank McM. Stanton.
Harry H. Taft.
James F. Tarwater.
John W. Taylor.
William D. Thornton.
Thomas Tonge.
Edgar G Tuttle.
John J. Vandemaer.
Henry A. Vezin.
Dr. Elwyn Waller.
William Shaw Ward.
Willard Warner, Jr.
William Watson.
Charles H. Wellman.
William Weston.
Howard Wierum.
John Wilkes.
Elwood J. Wilson.
T. R. Woodbridge.

PAPERS.

The Cycle of the Plunger-Jig.

BY ROBERT H. RICHARDS, BOSTON, MASS.

(Pittsburgh Meeting, February, 1896.)

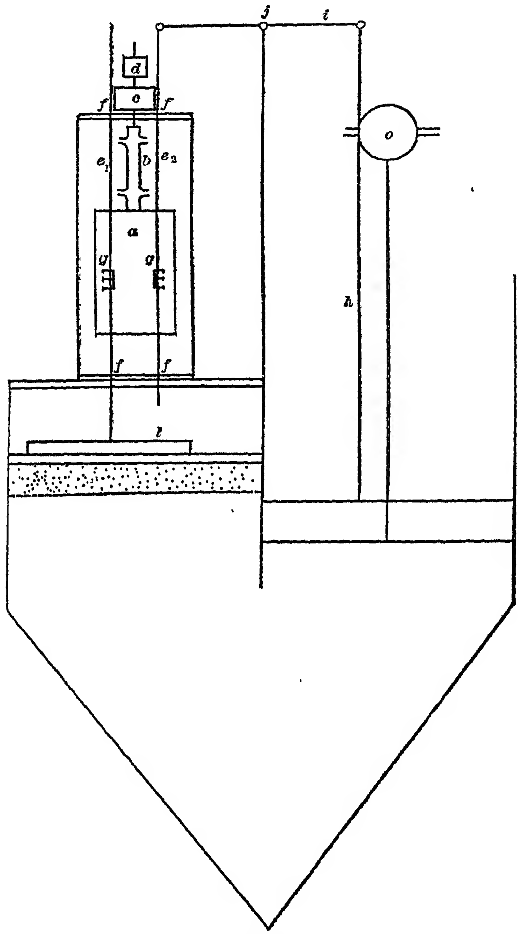
IN the discussion of my paper on "Close Sizing Before Jigging," * Mr. Louis remarks: † "What we really need to know as the basis of any consistent theory (of jigging) is what occurs during each hundredth of a second."

It is clear that for the further study of the laws involved, a jig-tester is needed which will give curves representing the amount, rate and kind of motion taken on by the plunger, the water and the sands. In pursuance of this idea the author has designed a jig-indicator for drawing curves illustrating the action of plunger-jigs. The indicator (see Fig. 1) consists of a cylinder, *a*, of brass, 8 inches long, 6 inches diameter, with its axis vertical, suspended from and continuous with a bicycle ball-bearing, *b*, to eliminate friction. The cylinder is rotated by clockwork, *c*, with a wind-wheel escapement, *d*, and can be made to revolve from 3 to 25 times per minute, according to the size of the wind-wheel. Upon the surface of the cylinder is wrapped a piece of paper, 8 inches wide and 20 inches long, upon which the curves are drawn. Two vertical recording-rods, e_1, e_2, running in anti-friction roller-guides, *f f f f*, with pencils, *g g*, attached, and provided with devices for throwing them into gear and out of gear, serve to record vertical motion upon the paper. The abscissa of the curve represents time, the ordinate represents movement up and down. The curve of the plunger is obtained by a vertical rod, *h*, held down on the top of the plunger and transmitting the oscillations of the latter by a horizontal beam, *i*, oscillating upon its center point, *j*, as a pivot to one of the recording-rods, e_2. When the plunger moves down, its pencil records an upward motion, and *vice versa*. To get the curve of the surface of the water a slab of

* *Trans.*, xxiv., 409. † *Id.*, p. 919.

cork, l, 10 inches square and 1 inch thick, is floated upon it and attached to the other recording-rod, e_1, direct. Here the motion of the pencil is the same as that of the water. The curve of the

FIG. 1.



Jig-indicator.

top-layer of quartz-sand is obtained by placing on the sand a piece of sieve-cloth a little finer in mesh than the grains of quartz. This pulsates up and down with the sand, and when connected to one of the recording-rods, e_1, gives the curve of the quartz. The curve of the ore-bed which underlies the

quartz may be obtained by sinking the piece of sieve-cloth to the desired depth and attaching it to the recording-arm, e_1.

Four curves are thus obtained: that of the plunger, marked P on the diagrams; that of the water, marked W; that of the quartz or rock, marked R; and that of the ore-bed, marked O. (The diagrams are shown in Figs. 2 to 41 inclusive, all of which are of natural size.)

In the actual performance of the tests two curves only are taken at a time—namely, those of the plunger and the water; those of the plunger and the quartz or rock; and, finally, those of the plunger and the ore-bed. The diagrams have been constructed by combining these drawings. In this operation it was necessary to have some standard by which, for example, the rock-curve and the ore-curve could be superposed upon the water-curve. Such a standard was found in the parallel plunger-curves that went with each. Again, the superposed curves would often come longer or shorter than the water-curve. In that case a new curve was sketched, making the deviation on the abscissa proportional to the distance from the starting-point.

Curves have been obtained at the following mills; the Anaconda, Montana (argentiferous sulphide copper-ore); the Calumet and Hecla, Michigan (native copper); the Revenue Tunnel, Colorado (argentiferous galena); the Smuggler, Colorado (argentiferous galena); the Old Jordan and Galena, Utah (auriferous pyrites); the Colorado Mining and Smelting Co., Montana (argentiferous sulphide copper-ore); the Bunker Hill and Sullivan, Idaho (argentiferous galena); the Boston and Montana Silver and Copper Mining and Smelting Co., Montana (argentiferous sulphide copper-ore); and the Butte and Boston Mining Co., Montana (argentiferous sulphide copper-ore).

Thanks are due to the officers of these companies for their kindness in aiding me to make these tests and to gain the other facts here presented.

The cards at the Smuggler mill were taken by Mr. S. I. Hallett; the rest by the author.

The jigs divide themselves into two main classes, the plain excentric Harz jigs and the accelerated jigs, including the crank-arm, the sliding-block and the Collom jigs.

Before drawing conclusions as to the meaning of the curves

here shown, it will be well to recall briefly certain facts demonstrated in my paper on "Close Sizing Before Jigging."*

It was there shown that three laws of jigging had been advanced by various authorities, and a fourth law was advanced by the author.

1. The law of free-settling particles (the equal-settling particles of Rittinger).
2. The law of hindered-settling particles (the interstitial currents of Munroe).
3. The law of acceleration (Rittinger).
4. The law of suction (advanced by the author).

The results obtained on free-settling particles are embodied in the following table, corresponding with Table VII. of the paper above cited:

TABLE I.—*Free-Settling Factors, or Multipliers for Obtaining the Diameters of Quartz which will be Equal-Settling with the Mineral Specified when Settling Freely in Ample Water.*

	Velocity in inches per second.								
	1.	2.	3.	4.	5.	6.	7.	8.	9.
	MULTIPLIERS.								
Anthracite	.500	.352	.225	.213	....	....	....	....	....
Epidote	1.57	1.35	1.05	1.13	1.50	1.61	1.56	1.56	1.47
Sphalerite	....	1.46	1.05	1.17	1.62	1.64	1.68	1.66	1.56
Pyrrhotite	....	1.73	1.29	1.48	2.00	2.22	2.26	2.13	2.08
Chalcocite	....	1.90	1.47	1.62	2.07	2.28	2.41	2.44	2.17
Arsenopyrite	....	1.90	1.57	1.89	2.42	2.56	2.72	2.84	2.94
Cassiterite	....	2.11	1.79	2.00	2.73	2.93	3.03	3.05	3.12
Antimony	....	2.71	2.00	2.00	2.73	2.93	3.03	2.98	3.00
Wolframite	....	2.71	1.83	2.07	2.86	3.04	3.21	3.28	3.26
Galena	....	2.71	1.83	2.26	3.00	3.42	3.65	3.76	3.75
Copper	....	2.71	2.00	2.36	3.00	3.20	3.58	3.76	3.75

Example.—If a compact particle of galena, falling freely in water, settles 7 inches per second, the particle of quartz of the same shape that will settle at the same rate will be approximately 3.65 times the diameter of the galena.

The law of hindered-settling deals with particles settling in an upward current, when they are in a mass and are bumping against and hindering one another; in fact, in the condition of

* *Trans.*, xxiv., 409.

quicksand. Table II., abstracted from "Close Sizing Before Jigging," gives the author's factors for particles that would settle with, and be adjacent to, each other under these conditions.

TABLE II.—*Hindered-Settling Factors, or Multipliers for Obtaining the Approximate Diameter of the Particles of Quartz Settling in a Mass of Grains that will be in Equilibrium with, and Adjacent to, the Mineral Specified:*

Particles of quartz	will have	8.6	times the diameter of	particles of	copper.
"	"	5.8	"	"	galena.
"	"	5.2	"	"	wolframite.
"	"	4.9	"	"	antimony.
"	"	4.7	"	"	cassiterite.
"	"	3.7	"	"	arsenopyrite.
"	"	3.1	"	"	chalcocite.
"	"	2.8	"	"	pyrrhotite.
"	"	2.1	"	"	sphalerite (blende).
"	"	1.6	"	"	epidote.
" anthracite	"	5.6	"	"	quartz.

Example.—If mixed grains of quartz and galena are allowed to arrange themselves in a mass, hindering and elbowing one another all the time, in an upward-moving current of water, the quartz particles, when equilibrium is reached, will be found to have approximately 5.8 times the diameter of the particles of galena poised alongside of them.

I use here the terms "free-settling factors" and "hindered-settling factors" instead of the phrases "equal-settling factors" and "interstitial-current factors," respectively used in my former paper, because I believe the new phrases are more expressive of the conditions most prominent in each case.

The third law, Rittinger's law of acceleration, deals with the behavior of particles suddenly changing their condition from rest to motion. Rittinger demonstrated that if a small particle of galena were adjacent to a larger particle of quartz, the galena would reach its maximum speed of settling sooner than the quartz, and that this fact would aid in separating the galena particles from the quartz.

The fourth law of suction, advanced by the author, is well shown by the figures of Table III., abstracted from Table XLIII. of the paper on "Close Sizing Before Jigging,"* where

* *Trans.*, xxiv., 472.

a series of tests in the jigging of blende and quartz together was reported. In the first, the blende is of the same size as the quartz; in the others, the diameter of the blende is diminished in each successive test. The measure of easy or difficult jigging is the number of pulsions needed to make a separation. The amount of the separation, as judged by the eye, is inserted in the table.

TABLE III.—*Jigging Quartz and Sphalerite (Blende).*

	In.	In.	In.	In.	In.	In.
Diameter of quartz	0.0683	0.0683	0.0683	0.0683	0.0683	0.0683
" blende	0.0683	0.0429	0.0262	0.0195	0.0095	0.0042
Series 1, with Strong Suction.						
Pulsions needed for separation	2129	1676	1759	297	208	288
Per cent. of blende brought down	96	95	95	95	99	99
Series 2, with Mild Suction.						
Pulsions needed for separation	306	838	846	1382	1729	∞
Per cent. of blende brought down	99	99	100	98	97	0
Series 3, with No Suction.						
Pulsions needed for separation	147	202	496	∞	∞	∞
Per cent. of blende brought down	98	95	50*	0	0	0

* No more would come down.

For the method of making the three series of tests, the first with strong suction, the second with mild suction, and the last with no suction, the reader is referred to the paper cited.

The law of suction seems to be that jigging is greatly hindered by strong suction where the two minerals are nearly of the same size, the quickest and best work being then done with no suction;* but that when the two minerals differ much in size of particles, the quartz being the larger, strong suction is not only a great advantage, but may be necessary to get any separation at all.† My experiments have indicated an approximate boundary between grains that are helped and those that are hindered by suction. Namely, if the diameter of the quartz-particles is equal to or greater than 3.52 times the diameter of

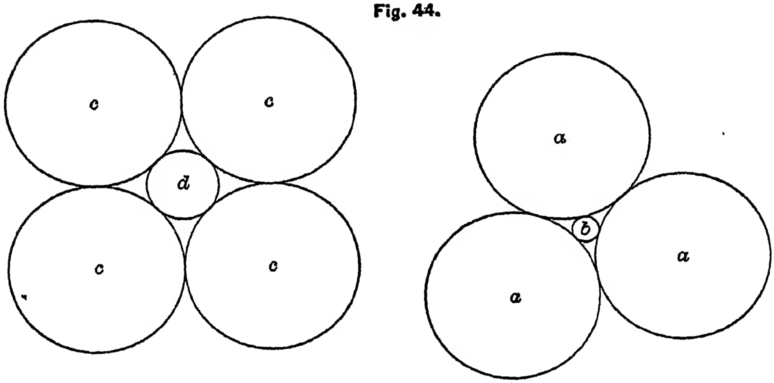
* Compare the first column of Series 3 with that of Series 1.

† Compare the fourth column of Series 1 with that of Series 3.

the other mineral-particles, then separation is helped by suction; if less, separation is hindered. This value, 3.52 (obtained by dividing 0.0683 by 0.0195), is approximate only, and it will differ with the fracture of the quartz under consideration; if the quartz grains are much flattened it will have a large value. The grains are of such irregular shapes that geometrical analogy fails to help in obtaining a value; for example, in Fig. 44 the diameters of the spheres of quartz, *a a a*, will be 6.50 times that of the ore, *b*, when the ore can slip through between the quartz, while the diameters of the quartz spheres, *c c c*, will be only 2.44 times that of the ore, *d*.

Table IV. contains the measurements and computations made

Fig. 44.



upon the jigs, and upon the cards obtained from them. The four elements that are new are:

1. The upward velocity of the water on the pulsion-side of the curve.
2. The percentage of time devoted to pulsion.
3. The percentage of time devoted to return.
4. The percentage of time devoted to suction.

The first of these is a rather weak determination, as the cards, even of an excentric jig, did not always show exactly the same slope on both sides. They are, however, accurate enough (since, where the error exists, it makes the figure a little too high) to demonstrate the fact that the law of free-settling particles has no bearing upon jigging, unless it may apply in the case of a few stray, floating grains on the surface of the finest jigs; for, on the one hand, the current was in no case strong

Table IV.—*Measurements of the Jigs and Curves.*

A. *Plain Harz Excentric Jigs.*

Figure No.	Name of Mill.	Plunger. Throw in inches.	Plunger. Throws per minute.	Plunger. Seconds per throw.	Product Fed. Passing through sieve-hole of diameter of	Product Fed. Resting on sieve-hole of diameter of	Product Fed. From	Sieve of Jig. Mesh or Diameter of hole.	Sieve of Jig. Area of Sieve in the clear.	Water Velocity during Pulsion. Inches per second.	Stroke Divided as Follows: Pulsion.	Stroke Divided as Follows: Return.	Stroke Divided as Follows: Suction.
									Inches.		Per ct.	Per ct.	Per ct.
2	Butte and Boston	4.75	140	0.429	2⅛ in.	1½ in.	Trommel.	⅜ in. rou'd.	24 by 48	13.92	34.0	25.0	41.0
3	" "	4.17	140	0.429	1½ in.	15 mm.	"	⅜ "	24 " 48	10.55	33.3	23.2	43.4
4	" "	2.20	140	0.429	15 mm.	8¼ mm.	"	4-mesh.	17 " 30	9.68	30.4	24.0	45.6
5	" "	2.00	160	0.375	8¼ mm.	4½ mm.	"	4- "	15 " 31	7.92	32.7	27.9	39.4
6	" "	1.11	180	0.333	4½ mm.	....	1st spigot of hydraulic separator.	5- "	15 " 31	4.74	36.5	37.6	25.8
7	" "	1.19	180	0.333	4½ mm.	....	2d spigot ditto.	6- "	15 " 31	3.87	42.8	27.0	30.2
8	" "	.48	200	0.300	4½ mm.	....	3d spigot ditto.	8- "	15 " 31	2.15	38.2	41.8	20.0
9	" "	.28	210	0.286	4½ mm.	....	4th spigot ditto.	12- "	15 " 31	1.44	33.3	43.7	22.9
10	Boston and Montana	2.57	165	0.364	1½ in.	⅞ in.	Trommel.	⅜ in. sq.	21½ " 42	11.99	49.2	20.9	29.9
11	" "	1.77	174	0.345	⅞ in.	⅜ in.	"	3-mesh.	24 " 36	9.11	43.0	24.1	32.9
12	" "	1.05	180	0.333	5 mm.	2½ mm.	"	6- "	21 " 41	6.51	38.9	26.4	34.7
13	" "	.75	193	0.311	2½ mm.	....	1st spigot of hydraulic separator.	6- "	22 " 39	1.71	......	......	......
14	" "	.39	193	0.311	2½ mm.	....	4th spigot ditto.	10- "	22 " 39	6.79	41.4	32.8	25.8
15	Bunker Hill and Sullivan	1.45	150	0.400	7 mm.	5 mm.	Trommel.	8- "	15 " 30	4.88	47.9	27.4	24.7
16	" "	.68	162	0.370	3 mm.	....	1st spigot of hydraulic separator.	10- "	18 " 32	4.09	43.6	37.1	19.3
17	Colorado M. and S. Co.	1.05	160	0.375	7 mm.	4½ mm.	Trommel.	3- "	23 " 35	6.64	42.3	26.9	30.8
18	" " "	.57	210	0.286	3 mm.	....	Only spigot of hydraulic separator.	10- "	23 " 35	4.81	35.6	45.8	18.6
19	Old Jordan and Galena	.67	240	0.250	⅜ in.	¼ in.	Trommel.	2½ "	17 " 30	3.32	52.0	18.0	30.0
20	" " "	.20	400	0.150	⅛ in.	....	Only spigot.	8- "	17 " 30	2.24	44.8	20.7	34.5
21	Revenue Tunnel	.27	198	0.303	5/64 in.	....	1st spigot of hydraulic separator.	16- "	16 " 23	......	24.5	34.7	40.8

B. *Accelerated (Crank-Arm) Jigs.*

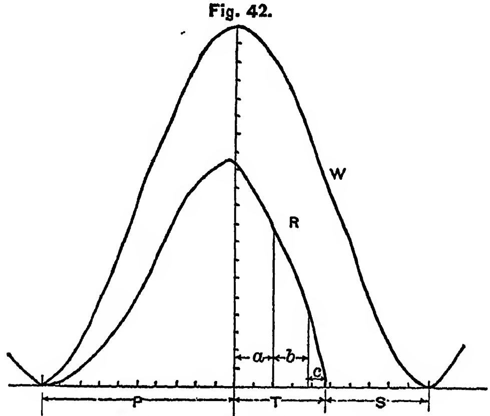
Figure No.	Name of Mill.	Plunger.			Product Fed.			Sieve of Jig.		Water Velocity during Pulsion. Inches per second.	Stroke Divided as Follows:		
		Throw in inches.	Throws per minute.	Seconds per throw.	Passing through sieve-hole of diameter of	Resting on sieve-hole of diameter of	From	Mesh or Diameter of hole.	Area of Sieve in the clear.		Pulsion.	Return	Suction.
									Inches.		Per ct.	Per ct.	Per ct.
22	Smuggler Mining Co.	2.42	96	0.624	40 mm.	25 mm.	Trommel.	8 mm.	18 by 34	9.66	28.1	23.7	48.2
23	" "	2.10	105	0.571	25 mm.	16 mm.	"	8 mm.	18 " 34	13.25	26.7	26.1	47.2
24	" "	1.42	121	0.496	16 mm.	12 mm.	"	8 mm.	18 " 29	10.40	28.6	30.3	46.1
25	" "	1.36	123	0.488	12 mm.	8 mm.	"	5 mm.	18 " 29	8.73	23.2	26.5	50.3
26	" "	1.18	132	0.455	8 mm.	5 mm.	"	5 mm.	18 " 29	11.76	29.7	33.7	36.6
27	" "	1.07	138	0.435	5 mm.	3½ mm.	"	2 mm.	18 " 29	10.70	22.6	31.1	45.9
28	" "	0.81	133	0.451	3½ mm.	2 mm.	"	1½ mm.	18 " 29	7.53	27.1	45.8	27.1
29	" "	0.70	133	0.451	3½ mm.	2 mm.	"	2 mm.	18 " 29	6.94	24.2	28.1	47.7
30	" "	0.51	135	0.444	2 mm.	....	1st spigot of hydraulic classifier.	5 mm.	18 " 29	4.45	27.0	46.8	26.2
31	" "	0.36	141	0.426	2 mm.	....	2d spigot ditto.	5 mm.	18 " 29	2.77	21.3	51.5	27.2
32	" "	0.40	163	0.370	2 mm.	....	3d spigot ditto.	2 mm.	18 " 29	4.36	40.9	59.1	0
33	" "	0.27	180	0.333	2 mm.	....	4th spigot ditto.	2 mm.	18 " 29	3.38	34.8	44.1	21.1
34	Revenue Tunnel	1.38	130	0.462	$\frac{11}{32}$ in.	$\frac{11}{64}$ in.	Trommel.	8-mesh.	17¾ " 24¾	8.95	18.5	34.6	46.9

C. *Accelerated Jigs (Collom).*

Figure No.	Name of Mill.	Throw in inches.	Throws per minute.	Seconds per throw.	Passing through sieve-hole of diameter of	Resting on sieve-hole of diameter of	From	Mesh or Diameter of hole.	Area of Sieve in the clear.	Water Velocity during Pulsion. Inches per second.	Pulsion.	Return	Suction.
35	Calumet and Hecla	0.75	128	0.469	3 in.	....	Residue in Leavitt Stamp.	6-mesh.	22 " 32	5.67	30.8	10.2	59.0
36	" "	0.51	134	0.448	$\frac{3}{16}$ in.	....	1st spigot of hydraulic separator.	8- "	22 " 32	4.80	32.6	44.2	23.2
37	" "	0.45	134	0.448	$\frac{3}{16}$ in.	....	2d hutch of 1st spigot jig.	12- "	22 " 32	4.83	22.0	46.3	31.7
38	Anaconda	1.00	130	0.462	$\frac{3}{16}$ by $\frac{3}{8}$ in.	....	1st spigot of hydraulic separator.	8- "	22 " 34	3.89	27.0	33.7	39.3
39	"	0.87	140	0.429	$\frac{3}{16}$ " $\frac{3}{8}$ in.	....	2d spigot ditto.	10- "	22 " 34	3.95	26.8	39.2	34.0
40	"	0.62	140	0.429	$\frac{3}{16}$ " $\frac{3}{8}$ in,	....	3d spigot ditto.	10- "	22 " 34	3.03	26.7	33.7	39.5
41	"	0.37	150	0.400	$\frac{3}{16}$ " $\frac{3}{8}$ in.	....	4th spigot ditto.	12- "	22 " 34	3.36	23.2	35.4	41.4

enough to lift the particles according to the free-settling law, and, on the other hand, the particles were not under free-settling conditions, even if the current had been strong enough.

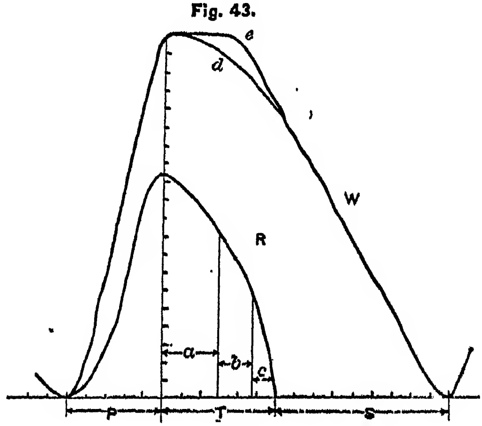
Fig. 42.



PLAIN HARZ ECCENTRIC JIG.
Ideal Diagram.

The curves of water and quartz will be of most interest, because it is from the quartz that we wish to catch the straying mineral, and it is by the water that we wish to catch it.

Fig. 43.



ACCELERATED JIG.
Ideal Diagram.

It will be seen upon inspection of the diagrams that the water-curve, W, and the quartz-curve, R, rise to their highest points and then descend, but that the quartz always gets down

before the water. We may say that there are three periods: 1. Pulsion; 2. Return; and 3. Repose or suction.

To aid in this discussion, two ideal diagrams (Figs. 42 and 43) have been drawn. In Fig. 43 two top-curves are given; the flat top, *e*, and the pointed top, *d*. A few of the diagrams (Figs. 26, 28, 33 and 34), indicate that the flat top is normal; but the remaining accelerated-jig curves speak for the pointed top.

The three periods named may be further discussed as follows:

Period I.—Pulsion, P, or upward movement. The water- and quartz-curves diverge, because the water is moving up faster than the quartz. Here the law of hindered settling is acting to bring the coarse galena below the quartz, and the fine galena adjacent to, and in equilibrium with, the proper size of quartz.

Period II.—Return downwards, T, which is divided into three periods:

(*a*) A moment when the quartz- and water-curves are converging, because the water is moving down faster than the quartz. Here the fine grains of galena acquire their maximum velocity downwards before the coarse grains of quartz; and hence Rittinger's law of acceleration is probably at work, helping the fine galena to get below the coarse quartz.

(*b*) A moment when the two lines of quartz and water are parallel, that is, a moment of relative idleness as to separation.

(*c*) A moment when the curves of quartz and water are diverging. The sand is falling faster than the water; hence the law of hindered settling is again at work.

Period III.—Suction, S. Here the sand reposes upon the sieve. The water-curve, W, is converging rapidly towards the horizontal quartz-line, R; therefore, the water is passing down through the sand at a high rate of speed. Here suction comes in to draw downwards through the interstices the small particles of galena which the law of hindered settling has placed adjacent to, and in equilibrium with, the larger particles of quartz.

Having thus observed the periods into which jigging is divisible, may we not conclude that a jig which is spending longer time upon the pulsion is doing the work of pulsion more thor-

oughly, and that a jig which devotes more time to suction is performing that special work more thoroughly? It should be noted here that the period of return counts to a greater or less extent as suction, particularly upon the jigs treating the finer sizes (as shown, for example, in Figs. 8, 9, 14, 21, 36 and 37). In all jigs the downward passage of the water is carrying into the hutch fine particles of ore that have already found their way into the ore-bed.

Looking at the jigs by classes, we see that the excentric jigs invariably spend more time upon pulsion than the accelerated jigs. Is it not fair to conclude that the excentric jigs are better adapted for treating sands which require the most pulsion? Such sands are the sized products from the trommel and the first spigot of the hydraulic classifier.

On the other hand, may not the long-protracted mild suction of the accelerated jig be the best adapted to the treatment of such products as require primarily suction for their separation —for example, the second spigot and following spigots of the hydraulic classifier? *

This may be the reason that the Collom jig has found so great favor at Lake Superior and at Anaconda, where all the jigging is done upon true hydraulic-separator products, except the first sieve of the first jig.

We should, however, bear in mind that the somewhat harsh suction of the excentric jig can be made milder by increasing the hydraulic water. This will diminish the hardening of the bed, but it cannot lengthen the time of suction, so as to secure the condition as presented in this particular by the accelerated jigs.

The two extreme suggestions arising from a contemplation of these curves are:

1. That on closely-sized products an accelerated jig should be used, run *backwards*, to lengthen out the pulsion-period, which is the only period that does any work; and

2. That the accelerated jig should be run *forward* on the spigot-products of the hydraulic separator, to increase the period of suction.

There are in the way of the first suggestion two difficulties,

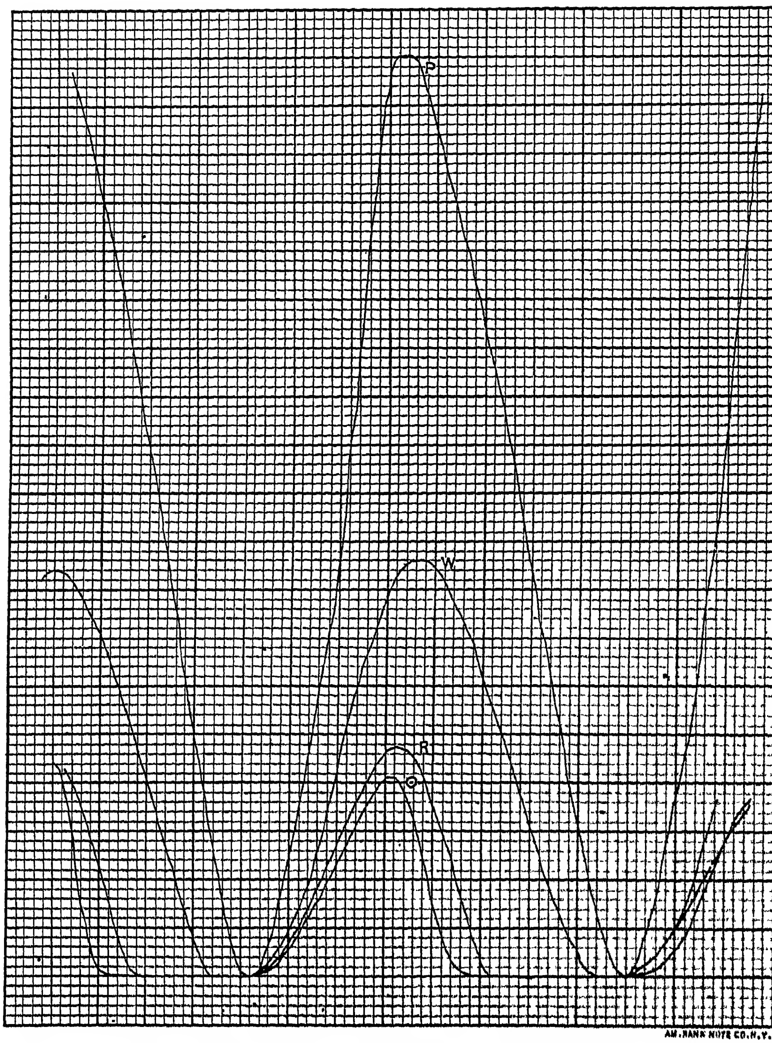
* Certain of these (for example, Figs. 30, 31 and 36, do not show this effect, for the reason, probably, that much hydraulic water was used).

either of which may cancel the advantage: First, the violent downward motion of the quick return will tend to "blind up" the sieve; and, secondly, the same action will tend to pulverize a soft mineral like galena.

I am aware that in southwest Missouri blende is jigged, all sizes together, from ½-inch downwards, upon a 6-sieve Harz jig, and that by using a moderately coarse sieve on the jig, and not allowing the bed to be too deep, and by employing a very strong suction, a concentration is effected which is considered commercially a success. It seems to me, however, probable that if the price of zinc-blende were not so low, the principles of close sizing would be adopted, in this, as in other cases, resulting in closer saving. In that case the principles discussed in this paper would be found essential to the greatest success.

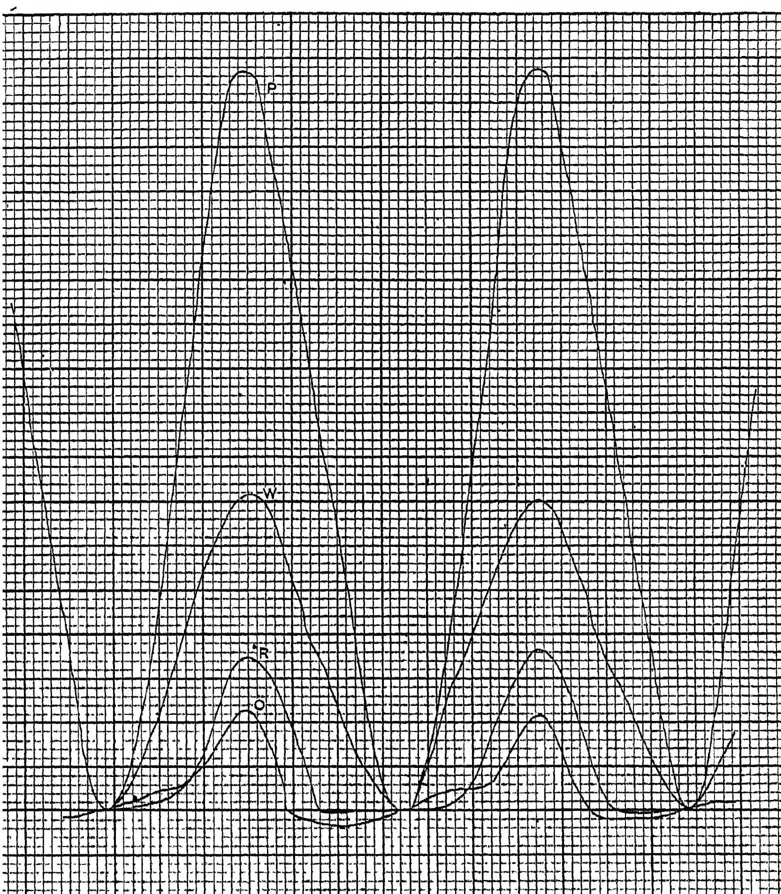
Nor am I ignorant of the wide difference between theoretical speculation and commercial operation, in which it often happens that some small, unnoticed need of practice undoes a beautiful theory, rendering it unsuited for adoption. The suggestions offered in this paper are therefore put forward simply as ideas which appear to have merit and to be worthy of further study and test.

FIG. 2.



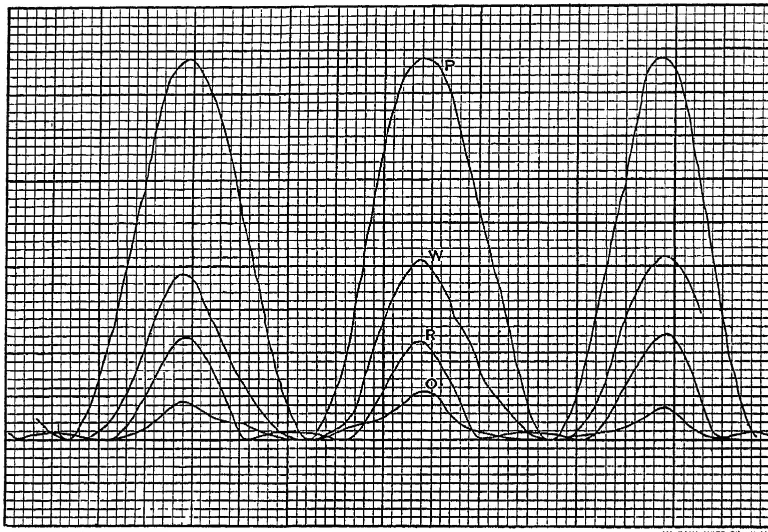
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains, $2\frac{1}{8}$ to $1\frac{1}{2}$ inches; throw of plunger, $4\frac{3}{4}$ inches; throws per minute, 140; area of plunger, 24 by 48 inches; area of sieve, 24 by 48 inches; mesh of sieve, round hole, $\frac{3}{8}$ inch diameter.

Fig. 3.



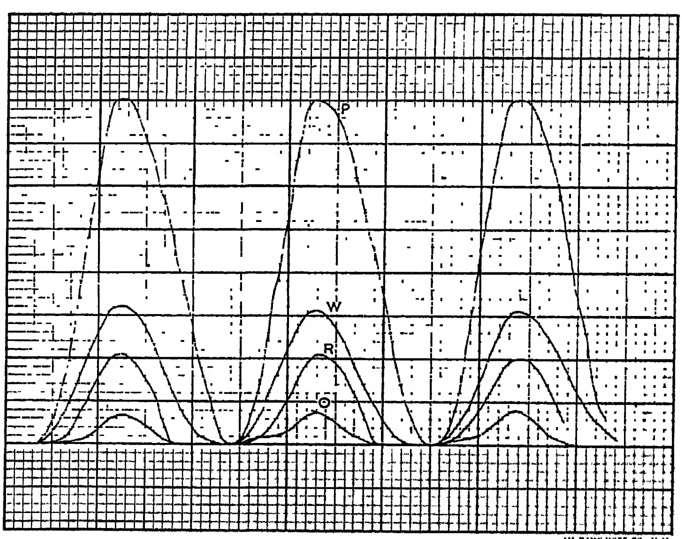
Butte and Boston Mining Co. (Argentiferous Sulphide Copper ore.) Size of grains, 1½ inches to 15 mm.; throw of plunger, 4.17 inches; throws per minute, 140; area of plunger, 24 by 48 inches; area of sieve, 24 by 48 inches; mesh of sieve, round hole, ⅝ inch diameter.

FIG. 4.



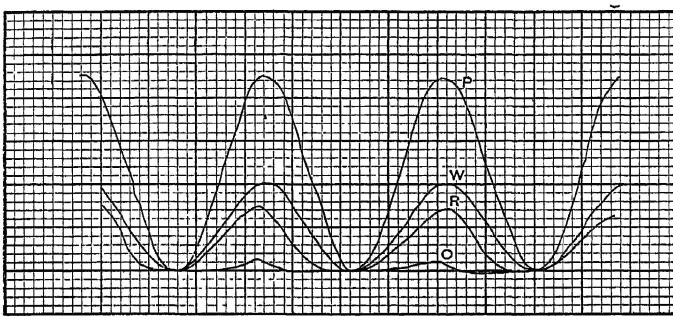
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains, 15 mm. to 8½ mm.; throw of plunger, 2.20 inches; throws per minute, 140; area of plunger, 17 by 30 inches; area of sieve, 17 by 30 inches; mesh of sieve, No. 4.

FIG. 5.



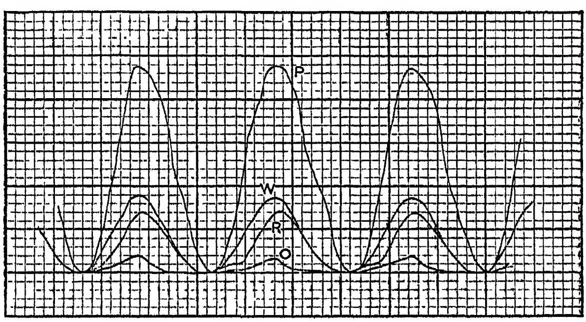
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains, 8½ to 4½ mm.; throw of plunger, 2 inches; throws per minute, 160; area of plunger, 15 by 31 inches; area of sieve, 15 by 31 inches; mesh of sieve, No. 4.

FIG. 6.



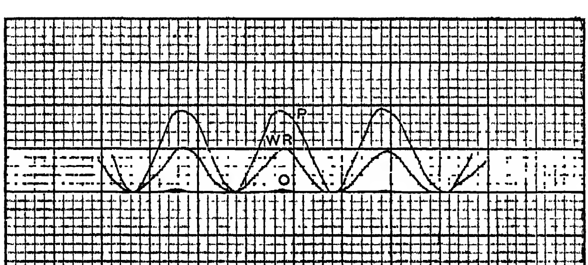
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains (first spigot of separator), 4½ mm. to 0; throw of plunger, 1.11 inches; throws per minute, 180; area of plunger, 15 by 31 inches; area of sieve, 15 by 31 inches; mesh of sieve, No. 5.

FIG. 7.



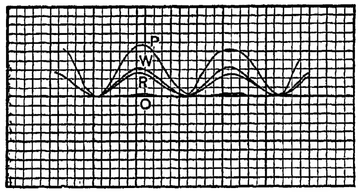
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains (second spigot of separator), 4½ mm. to 0; throw of plunger, 1.19 inches; throws per minute, 180; area of plunger, 15 by 31 inches; area of sieve, 15 by 31 inches; mesh of sieve, No. 6.

FIG. 8.



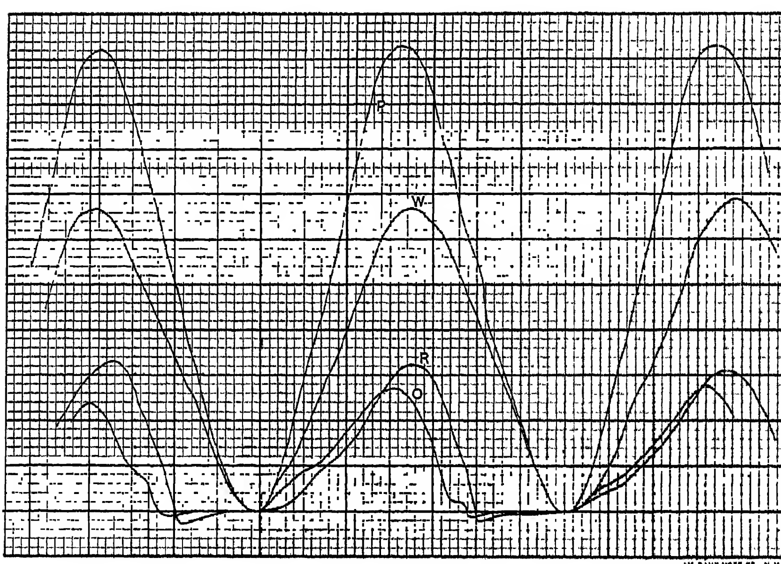
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains (third spigot of separator), 4½ mm. to 0; throw of plunger, 0.48 inch; throws per minute, 200; area of plunger, 15 by 31 inches; area of sieve, 15 by 31 inches; mesh of sieve, No. 8.

FIG. 9.



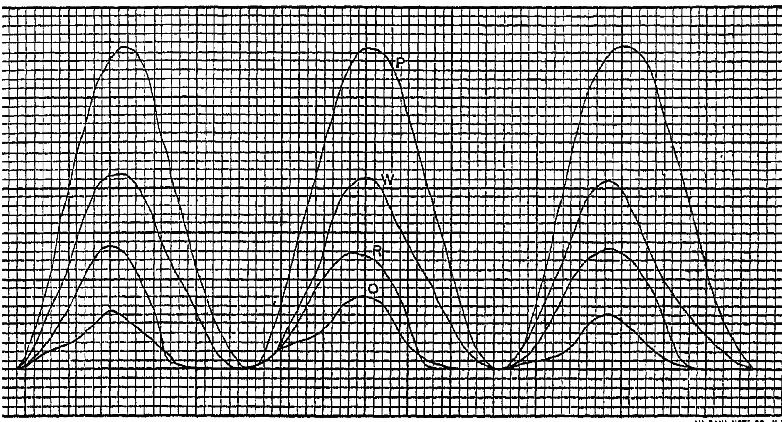
Butte and Boston Mining Co. (Argentiferous Sulphide Copper-ore.) Size of grains (fourth spigot of separator), 4½ mm. to 0; throw of plunger, 0.28 inch; throws per minute, 210; area of plunger, 15 by 31 inches; area of sieve, 15 by 31 inches; mesh of sieve, No. 12.

FIG. 10.



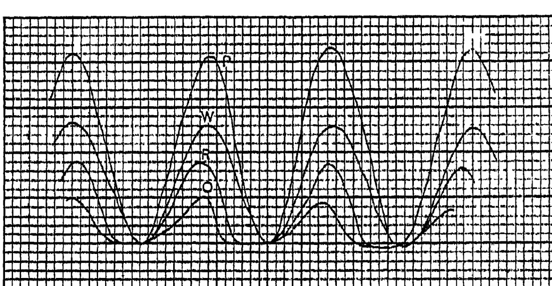
Boston and Montana Co. (Argentiferous Sulphide Copper-ore.) Harz Excentric Jig. Size of grains, 1½ to ⅞ inch; throw of plunger, 2.57 inches; throws per minute, 165; area of plunger, 22½ by 42 inches; area of sieve, 21½ by 42 inches; mesh of sieve, ⅝ inch square hole.

FIG. 11.



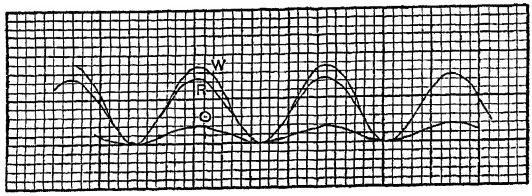
Boston and Montana Co. (Argentiferous Sulphide Copper-ore.) Size of grains, $\frac{7}{8}$ to $\frac{5}{8}$ inch; throw of plunger, 1.77 inches; throws per minute, 174; area of plunger, 24 by 36 inches: area of sieve, 24 by 36 inches; mesh of sieve, No. 3.

FIG. 12.



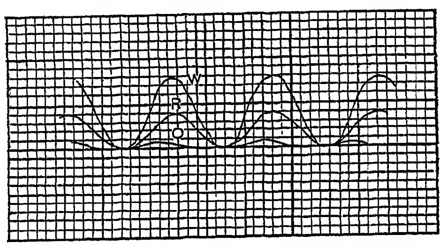
Boston and Montana Co. (Argentiferous Sulphide Copper-ore.) Size of grains, 5 to $2\frac{1}{2}$ mm.; throw of plunger, 1.05 inches; throws per minute, 180; area of plunger, 21 to 41 inches; area of sieve, 21 by 41 inches; mesh of sieve, No. 6.

FIG. 13.



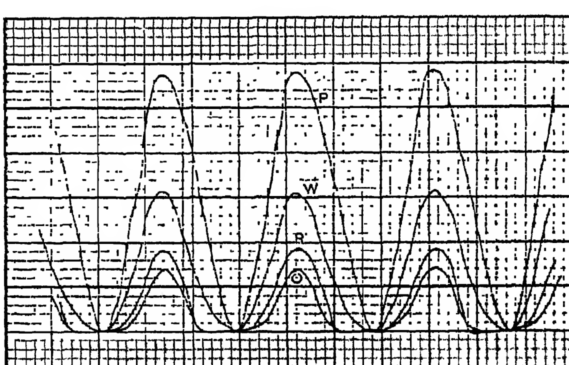
Boston and Montana Co. (Argentiferous Sulphide Copper-ore.) Evans' Excentric Jig. Size of grains (first spigot of separator), 2½ mm. to 0; throw of plunger, 0.75 inch; throws per minute, 193; area of plunger, 11 by 39 inches; area of sieve, 22 by 39 inches; mesh of sieve, No. 6.

FIG. 14.



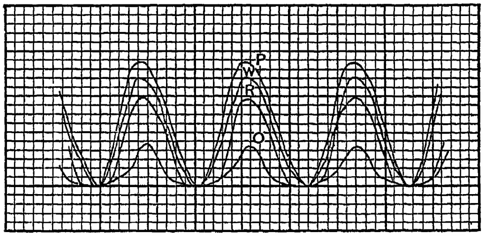
Boston and Montana Co. (Argentiferous Sulphide Copper-ore.) Evans' Jig. Size of grains (fourth spigot of separator), 2½ mm. to 0; throw of plunger, 0.59 inch; throws per minute, 193; area of plunger, 22 by 39 inches; area of sieve, 22 by 39 inches; mesh of sieve, No. 10.

FIG. 15.



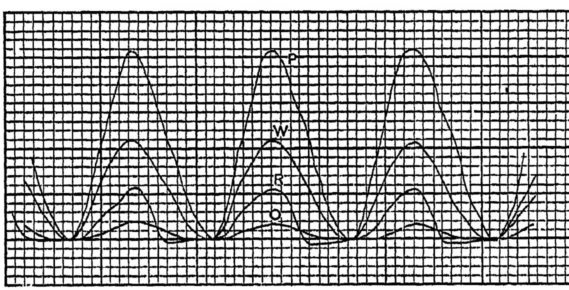
Bunker Hill and Sullivan Co. (Argentiferous Galena) Size of grains, 7 to 5 mm.; throw of plunger, 1.45 inches; throws per minute, 150; area of plunger, 17 by 32 inches; area of sieve, 15 by 30 inches; mesh of sieve, No. 8.

FIG. 16.



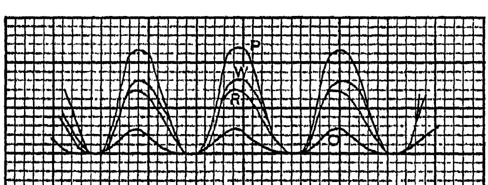
Bunker Hill and Sullivan Co. (Argentiferous Galena.) Size of grains (first spigot of separator), 3 mm. to 0; throw of plunger, 0.68 inch; throws per minute, 162; area of plunger, 19 by 34 inches; area of sieve, 18 by 32 inches; mesh of sieve, No. 10.

FIG. 17.



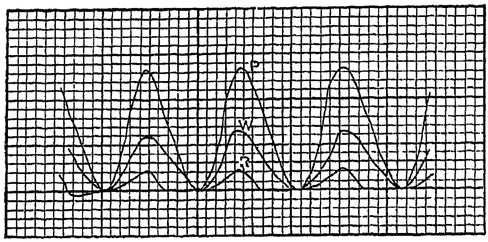
Colorado Mining and Smelting Co. (Argentiferous Sulphide Copper-ore.) Size of grains, 7 to 4½ mm.; throw of plunger, 1.05 inches; throws per minute, 160; area of plunger, 16 by 38 inches; area of sieve, 23 by 35 inches; mesh of sieve, No. 3.

FIG. 18.



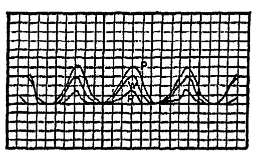
Colorado Mining and Smelting Co. (Argentiferous Sulphide Copper-ore.) Size of grains (only spigot of separator), 3 mm. to 0; throw of plunger, 0.57 inch; throw per minute, 210; area of plunger, 16 by 38 inches; area of sieve, 23 by 35 inches; mesh of sieve, No. 10.

FIG. 19.



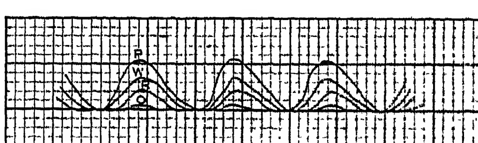
Old Jordan and Galena Works. (Auriferous Pyrites.) Size of grain, $\frac{4}{16}$ to $\frac{1}{4}$ inch; throw of plunger, 0.67 inch; throws per minute, 240; area of plunger, 17 by 30 inches; area of sieve, 17 by 30 inches; mesh of sieve, No. $2\frac{1}{2}$.

FIG. 20.



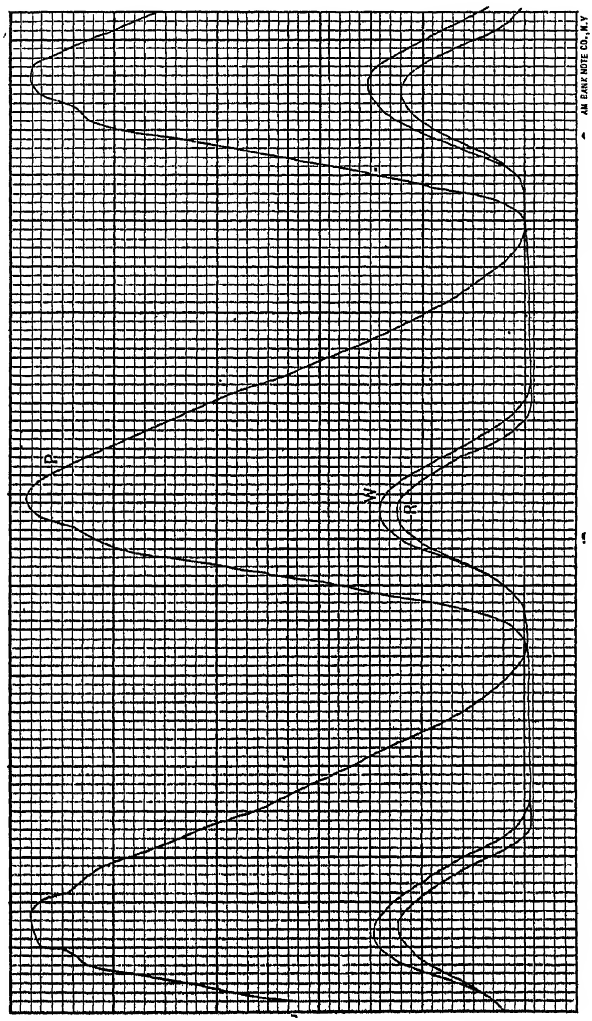
Old Jordan and Galena Works. (Auriferous Pyrites.) Size of grains (spigot of separator), $\frac{1}{8}$ inch to 0; throw of plunger, 0.20 inch; throws per minute, 400; area of plunger, 17 by 30 inches; area of sieve, 17 by 30 inches; mesh of sieve, No. 8.

FIG. 21.



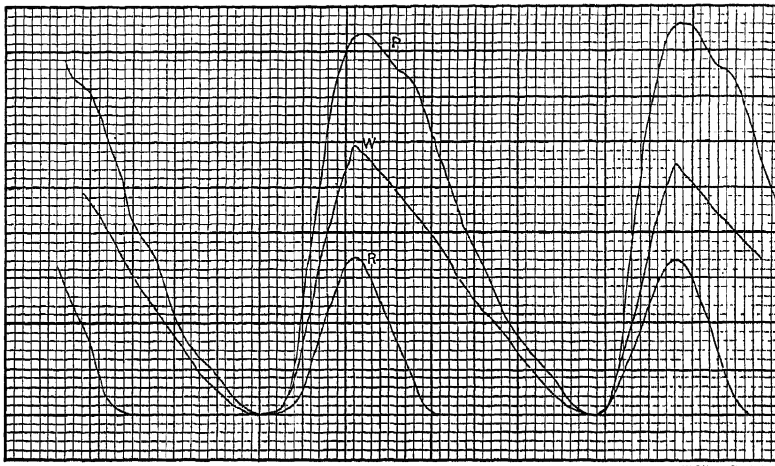
Revenue Tunnel. (Argentiferous Galena.) Plain Excentric Jig. Size of grains (first spigot of separator), $\frac{5}{64}$ inch to 0; throw of plunger, 0.27 inch; throws per minute, 198; area of plunger, $17\frac{3}{4}$ by $24\frac{3}{4}$ inches; area of sieve, 16 by 23 inches; mesh of sieve, No. 16.

Fig. 22.



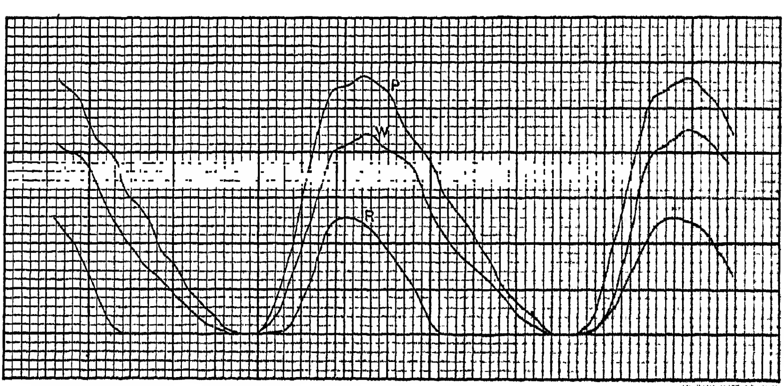
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 40 to 25 mm.; throw of plunger, 2.42 inches; throws per minute, 96; area of plunger, 18 by 30 inches; area of sieve, 18 by 34 inches; mesh of sieve, 8 mm. square hole.

FIG. 23.



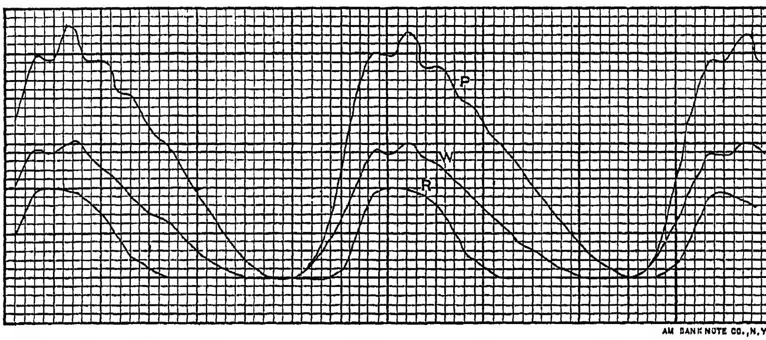
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 25 to 16 mm.; throw of plunger, 2.10 inches; throws per minute, 105; area of plunger, 18 by 30 inches; area of sieve, 18 by 34 inches; mesh of sieve, 8 mm. square hole.

FIG. 24.



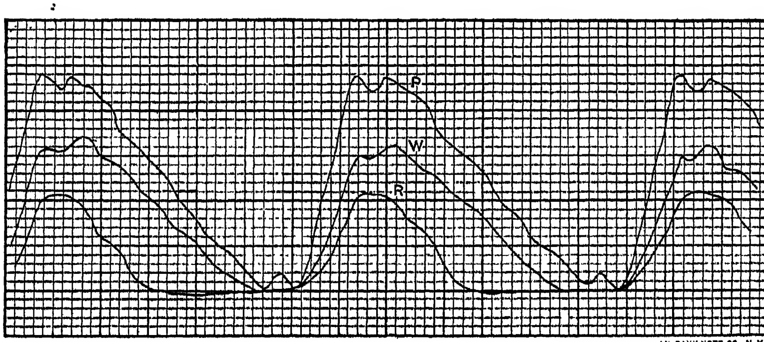
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 16 to 12 mm.; throw of plunger, 1.42 inches; throws per minute, 121; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 8 mm. square hole.

FIG. 25.



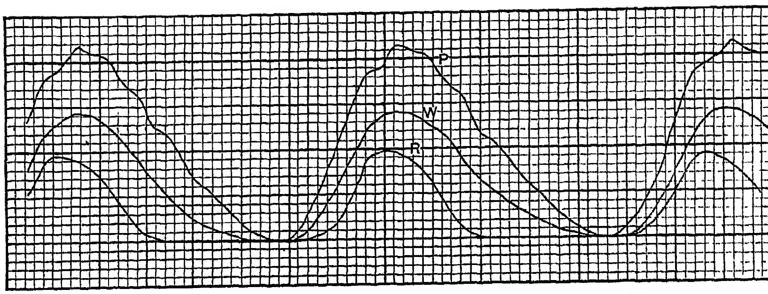
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 12 to 8 mm.; throw of plunger, 1.36 inches; throws per minute, 123; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 5 mm. square hole.

FIG. 26.



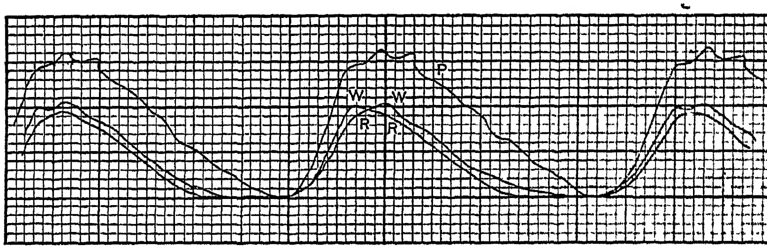
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 8 to 5 mm.; throw of plunger, 1.18 inches; throws per minute, 132; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 5 mm. square hole.

FIG. 27.



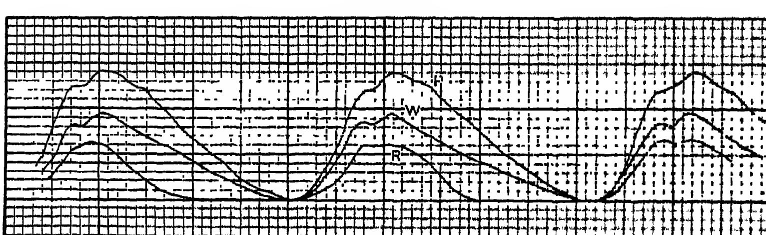
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 5 to 3½ mm.; throw of plunger, 1.07 inches; throws per minute, 138; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 2 mm. square hole.

FIG. 28.



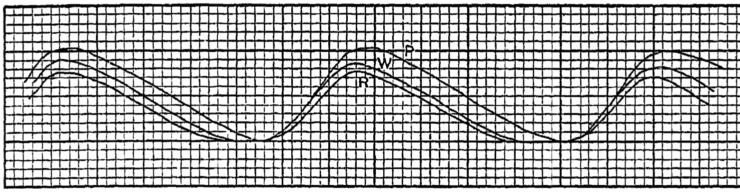
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 3½ to 2 mm.; throw of plunger, 0.81 inch; throws per minute, 133; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 1½ mm. square hole.

FIG. 29.



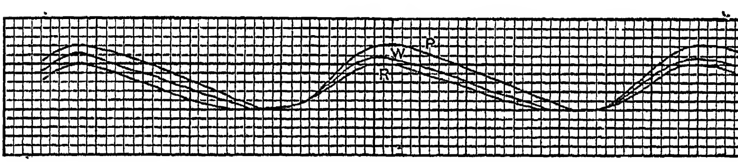
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, 3½ to 2 mm.; throw of plunger, 0.7 inch; throws per minute, 138; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 2 mm. square hole.

FIG. 30.



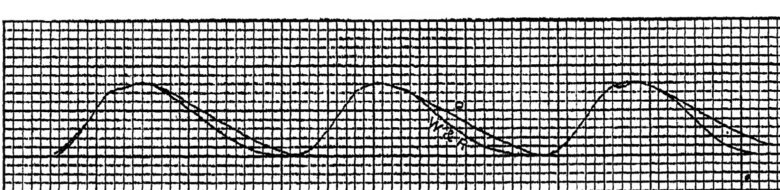
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains (first spigot of separator), 2 mm. to 0; throw of plunger, 0.51 inch; throws per minute, 135; area of plunger, 18 by 25 inches: area of sieve, 18 by 29 inches; mesh of sieve, 5 mm. square hole.

FIG. 31.



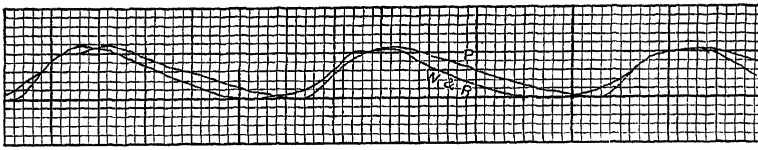
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains (second spigot of separator), 2 mm. to 0; throw of plunger, 0.36 inch; throws per minute, 141; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 5 mm. square hole.

FIG. 32.



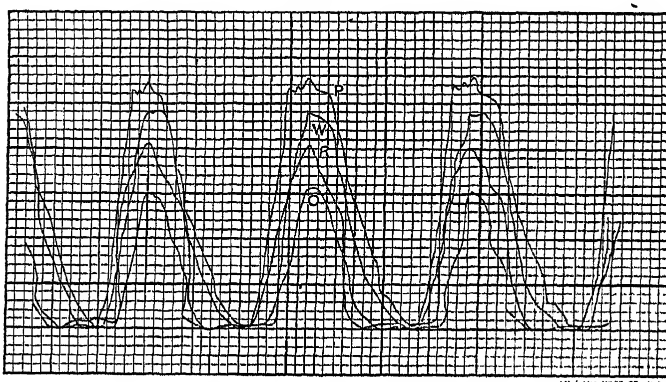
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains (third spigot of separator), 2 mm. to 0; throw of plunger, 0.4 inch; throws per minute, 163; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 2 mm. square hole.

FIG. 33.



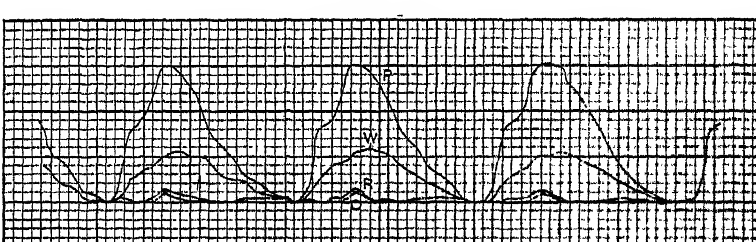
Smuggler Mill. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains (fourth spigot of separator), 2 mm. to 0; throw of plunger, 0.27 inch; throws per minute, 180; area of plunger, 18 by 25 inches; area of sieve, 18 by 29 inches; mesh of sieve, 2 mm. square hole.

FIG. 34.



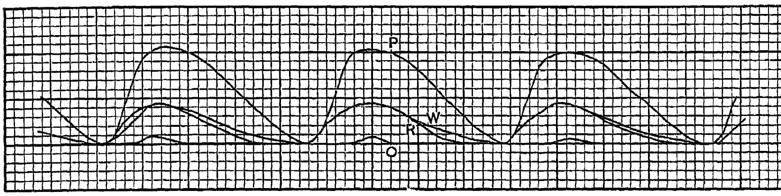
Revenue Tunnel. (Argentiferous Galena.) Crank-arm Accelerated Jig. Size of grains, $\frac{13}{32}$ to $\frac{21}{64}$ inch; throw, 1.38 inches; throws per minute, 130; area of plunger, $17\frac{3}{4}$ by $24\frac{3}{4}$ inches; area of sieve, 16 by 23 inches; mesh of sieve, No. 8.

FIG. 35.



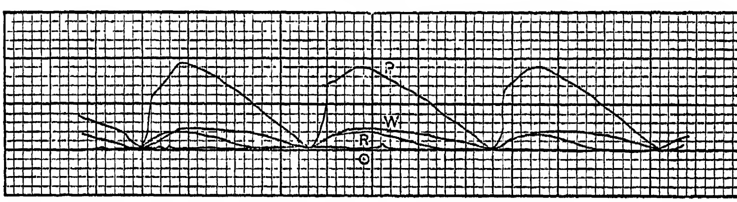
Calumet Mill of Calumet and Hecla Co. (Native Copper.) Cam-driven Spring-return Jig. Size of sand (residue in Leavitt Stamp-mortar), 3 inches to 0; throw of plunger, 0.75 inch; throws per minute, 128; area of plunger, 12 by 36 inches; area of sieve, 22 by 32 inches; mesh of sieve, No. 6.

FIG. 36.



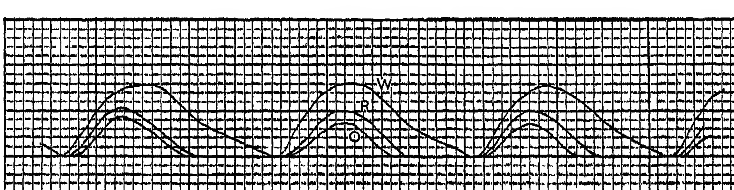
Calumet Mill of Calumet and Hecla Co. (Native Copper.) Collom Accelerated Hammer-driven Spring-return Jig. Size of sand (first spigot of separator, second sieve of jig), $\frac{3}{16}$ inch to 0; throw of plunger, 0.51 inch; throws per minute, 134; area of plunger, 22 by 17 inches; area of sieve, 22 by 32 inches; mesh of sieve, No. 8.

FIG. 37.



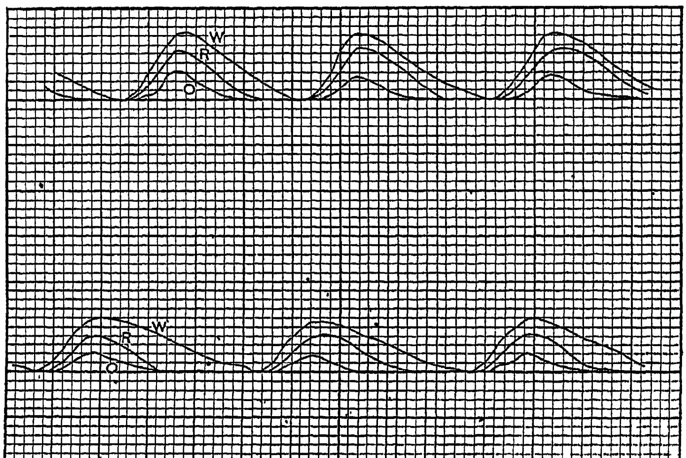
Calumet Mill of Calumet and Hecla Co. (Native Copper.) Collom Accelerated Hammer-driven Spring-return Jig. Size of sand (first spigot of separator, second sieve of finishing-jig), $\frac{3}{16}$ inch to 0; throw of plunger, 0.45 inch; throws per minute, 134; area of plunger, 22 by 17 inches; area of sieve, 22 by 32 inches; mesh of sieve, No. 12.

FIG. 38.



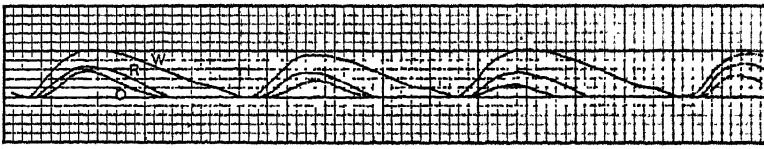
Anaconda Mill. (Argentiferous Sulphide Copper-ore.) Collom Accelerated Hammer-driven Spring-return Jig. Sand (first spigot of separator) has been through a screen with $\frac{3}{16}$ by $\frac{5}{8}$-inch slots; throw of plunger, 1 inch; throws per minute, 130; area of plunger, 22 by 17 inches; area of sieve, 22 by 34 inches; mesh of sieve, No. 8.

FIGS. 39 and 40.



Anaconda Mill. (Argentiferous Sulphide Copper-ore.) Collom Accelerated Hammer-driven Spring-return Jig. Fig. 39 (the upper diagram) represents the sand from the second spigot of separator; Fig. 40 from the third spigot; the throw of plunger is ⅞-inch in Fig. 39, and ⅝-inch in Fig. 40; throws per minute, 140; area of plunger, 22 by 17 inches; area of sieve, 22 by 34 inches; mesh of sieve, No. 10.

FIG. 41.



Anaconda Mill. (Argentiferous Sulphide Copper-ore.) Collom Accelerated Hammer-driven Spring-return Jig. Sand from fourth spigot of separator; throw of plunger, ⅝-inch; throws per minute, 150; area of plunger, 22 by 17 inches; area of sieve, 22 by 34 inches; mesh of sieve, No. 12.

The Accumulation of Amalgam on Copper Plates.

BY R. T. BAYLISS, MARYSVILLE, MONTANA.

(Pittsburgh Meeting, February, 1896.)

Although every mill-man of even limited experience in the amalgamation of gold-ores is probably aware that copper plates will in time become coated by the accumulation of gold-amalgam, it may be that many do not know to what extent this accumulation occurs in the treatment of ores dissimilar to those with which they are familiar, and under conditions foreign to those with which they have to contend in any given locality. The writer, therefore, feels relieved from the necessity of offering an apology for dealing with such an elementary subject, believing that certain facts in reference thereto, which have recently come under his personal observation, will be interesting to many, and possibly instructive to some persons engaged in the amalgamation of gold-ores.

The facts hereinafter stated were observed in the 50-stamp "combination"-mill, owned by the Montana Mining Company, Limited, of England, operating the Drum Lummon mine, situated at Marysville, Montana.

The material treated in this mill consists of quartz and ores of somewhat variable character, containing gold and silver in native form, as well as in chemical union with the sulphides of iron and copper. There is also present a little lead, some arsenic and antimony, and a little zinc. The last four minerals, however, are of irregular occurrence, and but sparingly distributed throughout the ore, and yield only a trace in the analysis of the average material in this mill. The free gold, which is in a condition of fine division, and rarely visible to the naked eye, carries silver as an alloy; and the native silver, which is only occasionally observed, is present in the richer ores in wire- and in leaf-form.

Of the two precious metals, the ore worked during the past three years and ten months, the period under review, contained about ½-ounce of gold, and from 7 to 12 ounces of silver, per ton of 2000 pounds.

The ore, after being crushed in the stamp-battery, and passed through a 30-mesh steel-wire screen, runs over amalgamated copper plates, and is then concentrated on Frue vanners, to the distributor of which is affixed a small copper plate for saving any amalgam escaping from the battery-plate; the tailings from the vanners being subsequently amalgamated in pans.

Amalgamation upon copper plates placed inside the mortar is not practiced in this mill, better results having been obtained by amalgamation on outside or apron-plates only. It is not the purpose of this paper to enter the wide field of discussion as to the relative merits of inside and outside amalgamation; but it may be observed in passing that an extended experiment upon the two methods proved that a higher percentage of saving was gained by amalgamation outside the battery, than by the use of inside plates in addition to the apron-plates.

The apron-plates in use in this mill are 54 inches wide and 8 feet long; made of ⅛-inch rolled copper plate, electro-plated with 1 ounce of silver to the square foot. They are set with a fall of 2¼ inches to the foot; are cleaned up once in twenty-four hours, and dressed once every two, three, or four hours, as may be deemed necessary by reason of their surface-condition, which varies according to the mineral contents of the ore submitted to them. In dressing, a weak solution of cyanide of potassium is used when necessary. The copper plates upon the Frue vanners are 46 inches wide, and 18 inches deep, made of ⅛-inch rolled copper plate, electro-plated with 1 ounce of silver to the square foot, and are subjected to the same treatment as the apron-plates.

At the daily clean-up, the surface of the apron-plates is first washed; then, the amalgam which has collected during the preceding twenty-four hours is slightly softened and loosened by sprinkling a small quantity of mercury over the plate and rubbing the surface with a cloth or whisk-brush, after which the amalgam is removed with a stiff rubber scraper, 4 inches wide, made out of rubber belting. In this process the daily accumulation of amalgam is removed as completely as possible without the application of a steel scraper, the use of which is not permitted except for the removal of blisters or any fixed impurity which may occasionally be found upon the plate.

The foregoing brief summary of the method employed in

this mill will be sufficient to explain the general conditions surrounding the amalgamation of the free gold contained in the ore, and the treatment to which the copper plates are exposed.

To return to the special subject indicated by the title of this paper: One of the copper plates employed in this mill was recently removed, after having been in continual service for three years and ten months. During this period, the battery of five stamps which it served crushed 14,942 tons of ore, yielding from amalgamation on this plate 6426 ounces of bullion, 541.5 fine in gold, and 443.9 fine in silver, giving a standard assay-value of $11.70 per ounce, and showing the recovery of $5.03 per ton of ore crushed. In fixing the total tonnage of ore passed over this plate, and the yield in bullion therefrom, the total tonnage and production of the mill during this period has been divided by ten. As all the stamps have been operated continuously, this proportion should correctly represent the duty performed by the particular plate under consideration.

The agreeable task of removing the amalgam known to have accumulated upon this plate (which was new when put in) was then undertaken, and the accumulation or scale was removed by striking the back and front of the plate with a light hammer, a small block of wood being used to deaden and distribute the blow. By this means the plate is slightly buckled, causing the amalgam to scale off, and leaving upon the original electro-plated surface but an insignificant film or layer, which is subsequently removed with a chisel or scraper.

Other methods, such as sweating with hot water and immersing the plates in chemical solutions, have been tried, but have proved unsatisfactory; and the process of buckling the plate, although a somewhat drastic and unscientific method, has been found to effect the recovery of the highest percentage of the accumulated amalgam. After this treatment, the plate is usually so damaged as to be unfit for further service; but, notwithstanding that every visible portion of amalgam has been removed, the copper plate is still found to carry a considerable quantity of gold and mercury; and upon being cut up and melted into a bar, the value of the gold- and copper-contents amounts to more than twice the value of a new plate; hence the substitution of new plates for old is an expenditure which can be viewed without concern.

The results obtained from the foregoing treatment of the particular copper plate here specified, were both interesting and of substantial value; the scale or accumulation of amalgam at the head of the plate being no less than 0.16 inch in thickness, and gradually decreasing to a bare $\frac{1}{16}$-inch at the lower end.

The total weight of the amalgam so recovered was 160 pounds avoirdupois, which, upon being retorted, yielded 60.5 pounds, or 38 per cent. of crude bullion, and produced after melting a gold bar weighing 866.1 ounces troy, which, upon assay, proved to have a total fineness of 993.9, being 431.4 fine in gold, and 562.5 fine in silver, and having a standard value of $9.63 per ounce, thus making the total value of bullion, recovered from the accumulated amalgam upon this one plate, no less than $8340.54.

It may be remarked, that the results recorded in this instance are not by any means exceptional. Other plates in this mill, placed in service at the same time as the one forming the subject of this paper, appear to be coated with an accumulation of amalgam of equal weight and value; and, moreover, within the writer's experience of other plates removed from this mill in past years, is an instance in which one plate yielded accumulated amalgam in excess of $11,000.

An examination of the facts observed in the course of cleaning this plate, and converting the amalgam into bullion, reveals one or two interesting features which appear to be worthy of record.

As stated above, the percentage of bullion in the amalgam obtained was 38 per cent.; whereas, the amalgam from the daily clean-up never contains more than 20 per cent. of bullion, and frequently not more than 10 per cent. This is, no doubt, due to the circumstance that the amalgam remaining upon the surface of the plate is subjected to greater compression than that which is cleaned off, and merely strained through canvas sacks.

The accumulated amalgam obtained from the plate in the shape of scale does not appear to suffer any visible alteration in form or size during the process of retorting. A piece $\frac{1}{8}$-inch thick and 1 inch square, will pass through the ordeal of retorting and emerge as crude bullion, having preserved, with-

out apparent loss, its original dimensions, and still retaining on its surface any ripple-marks or imperfections which it bore during its existence as amalgam.

A comparison of the fineness of bullion obtained from the accumulation of amalgam upon this plate with the average fineness of bullion produced from the daily clean-up of the plate, during its service of three years and ten months, of which one would suppose the accumulated amalgam to be a fair sample, shows that, to a striking degree, this is not the case.

The average fineness of bullion obtained from daily clean-ups during the period mentioned was Au, 541.5; Ag, 443.9; total, 985.4, whereas the fineness of bullion obtained from the accumulation upon this plate during the same period was Au, 431.4; Ag, 562.5; total, 993.9; from which it will be observed that although the total fineness of the latter, 993.9, is greater than the fineness of the former, 985.4, the gold-fineness is 110.1 less, and the silver-fineness 118.6 more.

That the total fineness of bullion from the accumulated amalgam should be higher than that obtained from the daily clean-ups is not a matter for surprise, since the amalgam gathered from the plates from day to day would naturally contain a higher percentage of impurity and base metal than the amalgam which adhered to the surface of the plate; but it does not clearly appear why the gold-fineness of the latter should be so much lower than that of the former, the difference being represented by a corresponding increase in the silver-fineness.

In explanation of this inconsistent feature, it has been suggested that the native silver contained in the ore has a greater tendency to accumulate upon the plates than the free gold, owing to its stronger affinity with the amalgamated surface. If theory furnishes any authority for such a statement, it is rudely disproved by actual experience in this particular instance; for test-samples of the accumulated amalgam from the head and tail of this plate prove the former to be .020 finer in gold than the latter, with a corresponding increase in the silver-fineness of the amalgam from the lower end of the plate.

Furthermore, the amalgam saved upon the copper plates forming the distributor of the Frue vanners invariably shows, upon assay, a lower gold- and a higher silver-fineness than the bullion recovered from the treatment of the same ore upon the

battery-plate; a sample of this vanner-amalgam yielding bullion assaying Au, 380.5; Ag, 602.0; total, 982.5. In other words, the silver, instead of showing a strong affinity with the amalgamated suface, gives evidence of a persistent tendency to escape amalgamation, as proved by the foregoing assays, which show that the gold-fineness of bullion is highest nearest to the battery, and gives place to a steadily-increasing silver-fineness as the amalgam is deposited upon the copper plates at greater distance from the battery-discharge.

The natural conclusion to be drawn from these facts and figures would justify the expectation that the bullion derived from the accumulation of amalgam on copper plates would be of equal (if not of higher) gold-fineness to that recovered from the daily clean-ups upon the plates during the period in which the accumulation was in progress. The foregoing facts, however, incontestably prove that such is not the case; and the purpose of this paper will be served if the facts herein presented, which have been prepared with due regard to accuracy, are the means of suggesting an explanation of this interesting inconsistency, which the writer most frankly confesses himself unable to supply.

Notes on the Handling of Slags and Mattes at Smelting-Works in the Western United States.

BY WILLIAM BRADEN, HELENA, MONTANA.

(Pittsburgh Meeting, February, 1896.)

It is obvious that the choice of the method to be employed in the handling of blast-furnace slags and mattes depends upon local facilities and conditions which may indicate as advisable some particular plan. As a result of the variety of such conditions hardly any two smelters use identically the same method, though they be located only a few hundred feet apart. For instance, of the several plants in Leadville, Colorado, each has a more or less distinctive system. The same is true of Salt Lake City and vicinity. In view of this it is evident that the existing diversity of practice cannot be accepted, *prima facia*, as evidence

of the lack of wisdom at any one works, and the following account of the methods pursued at several leading establishments in the West is offered, without purpose of criticism, as likely to furnish for all parties useful material for consideration.

The Boston and Montana Consolidated Copper and Silver Mining Company.—This company has large mines in Butte, Montana, whence the ore is shipped in cars to Great Falls, 170 miles distant, for treatment. Such ore as is suitably high in copper is smelted in blast-furnaces to a copper-matte, carrying from 50 to 55 per cent. of copper, which goes to the converters. These are as large as the average steel-converter, the minimum charge being about 3 and the maximum 8 tons. The crude ore, too low in copper to smelt direct, is concentrated, roasted in Brückner cylinders, and reduced to 50- to 55-per-cent. matte in reverberatory tilting-hearths. This matte, as well as the blast-furnace matte, is converted to "pig-copper," which is either refined electrolytically on the spot or shipped for refining elsewhere.

The works, located on the north bluff of the Missouri river, some two miles below the city of Great Falls, occupy a well-nigh ideal site for a large smelting-establishment, there being great water-power and a steady slope of about 1 foot in 5, more or less, from the edge of the river for several hundred feet back. Switches of the Great Northern Railway run through the works at the top and bottom with convenient grade for unloading of ore, coal, etc., and for the handling of empty cars.

Fuel for the "Wellman" and the "Taylor" gas-producers is supplied from the Sand Coulée coal-mines in the Great Falls district. Water, to be used under head, is pumped to large tanks on top of the bluff above the smelter.

The ultimate disposal of the slag at these works is effected by granulation and sluicing away; but the method may be better explained by considering the handling of the slag from three more or less distinct sources, namely, the blast-furnaces, the reverberatory tilting-hearths and the converters.

1. The molten material from the blast-furnaces is run continuously into a movable fore-hearth or settler, which is simply a rectangular box bound with cast-iron plates about 1 inch in thickness and lined with split fire-brick.

The matte (which is high in copper, thus assuring a clean slag) is tapped periodically, according to the amount being made,

from a tap-hole some 5 or 6 inches from the bottom of the fore-hearth, into an ordinary slag-pot. When this is full it is wheeled away a few feet and the matte is poured into moulds. When cool, being in slabs of from 2 to 3 inches in thickness, it is easily broken to suitable size.

The slag from the fore-hearth overflows continuously (except while the matte is being tapped) into a jet of water under pressure, and is sluiced into the river, where it is carried by the current.

2. At the tilting-furnaces, after a thorough fusion has taken place and the furnace is prepared for a separation of the slag and matte, it is tilted and a man with a long hoe skims the slag from the furnace into a trough, from which the slag is granulated and sluiced to the river like the blast-furnace slag above mentioned.

3. During the operation of skimming the tilting-hearths a converter is also being skimmed into a large ladle of 10 tons or more capacity. When the skimming of the converter and the tilting-hearth is finished both are swung to normal position and the ladle carrying the converter-skimmings is picked up by a 20-ton electric crane and carried to the tilting-hearth just skimmed, into which the slag is poured. By the cleansing on the "copper-bottoms," so to speak, of the tilting-hearth, lasting a few moments, I am told that the converter-slag is reduced from 2 to 2½ down to 0.6 per cent. of copper. After this cleansing the hearth is again tilted, and the slag is skimmed and granulated and sluiced off, as already described.

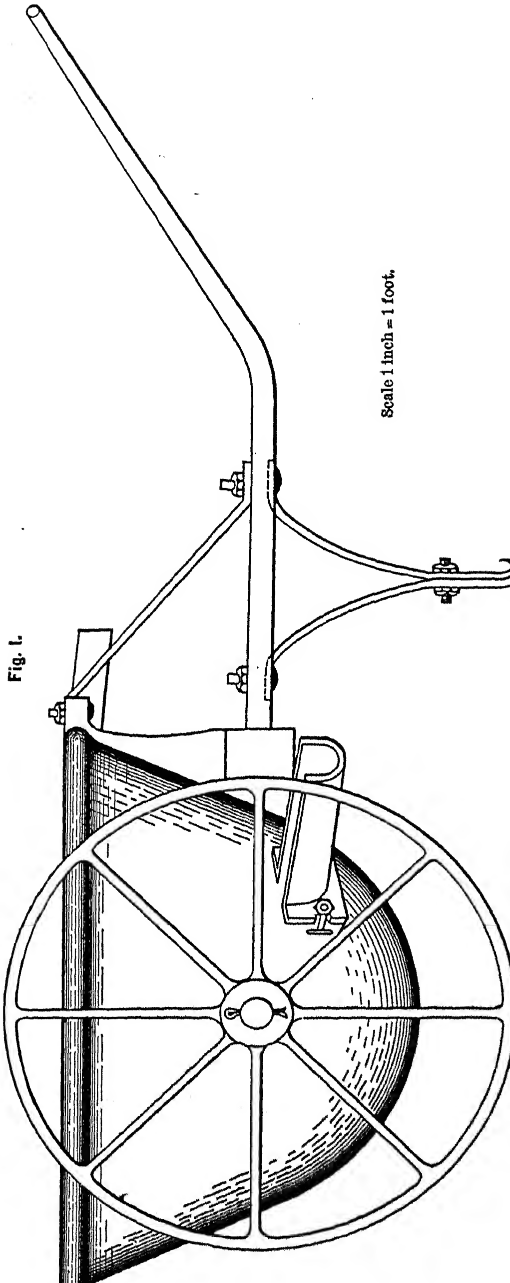
At other copper-works in Montana slag is handled from the blast-furnaces in much the same way where sufficient supply and head of water are available. But the slag from blast-furnaces more generally runs from the fore-hearth into an ordinary slag-pot, which is wheeled to the edge of the dump and emptied by tilting.

The United Smelting and Refining Company.—This company has near Great Falls, on the Missouri river, about 2 miles below the Boston & Montana Copper Works, above spoken of, works at which purchased ores are reduced to base-bullion. The sampling-works, roasters and ore-bins, blast-furnaces, engine-room and boiler-room are practically on the same level, which is that of a bluff above the river, where the furnaces are located, giv-

ing ample dumping-ground for the slag. The slag flows from the furnace into a settling-pot on the same principle and practically of the same dimensions as shown in Fig. 1 (which represents the exact form used at the Arkansas Valley Works, Leadville, Colo.). From this settling-pot the slag overflows into a Devereux-pot, which is simply an ordinary slag-pot, with a hole about 1 inch in diameter on the front side. In this case the hole is very low down, about 4 or 5 inches from the bottom of the pot. These Devereux pots, when filled, are pulled some 150 or 200 feet to the edge of the slag-dump and tapped by inserting a steel bar through the top crust and poking the same through the hole, which has previously been plugged with clay. Of course, when struck by the bar the plug of clay falls out, and the molten slag flows over the dump. If there happens to be any matte, it is saved in the portion of the pot beneath the hole. This is allowed to cool, and is then separated on the dump from the slag, by men with "culling-hammers." Usually there is between the slag and matte quite a perfect cleavage-plane, often characterized by crystallization. When a "settling-pot" has become filled with matte, a fresh one is substituted. The matte is allowed to cool and then broken up, as before described, and is ready for further manipulation. The advantage of the Devereux pot over the ordinary slag-pot, is connected with the possibility that a small quantity of matte may overflow with the slag from the settling-pot. In the use of the ordinary slag-pot everything except the crust adhering to the sides of the pot and a small amount of the top-crust would be thrown away over the dump; whereas, with the Devereux, if any matte is present, it is saved in the "button" below the hole. If, upon cooling, this "button" is seen to carry no matte, it is either thrown away, involving little additional expense (which is justified by the safeguard thus secured against loss), or it is re-smelted with the "crusts."

The Omaha and Grant Smelting Company.—At the works of this company, in Denver, Colorado, a very good system for handling slag and matte has been introduced. Slag is received from ten blast-furnaces. The molten material is tapped from the furnace into a slag-pot, which is wheeled away, and the slag is emptied as quickly as possible by tilting into a large pot constructed on the Devereux principle, which has previously been lowered into a cylindrical compartment or pit in the dump, so

Fig. 1.



Scale 1 inch = 1 foot.

Settling-pot, used at Arkansas Valley Works, Leadville, Colo.

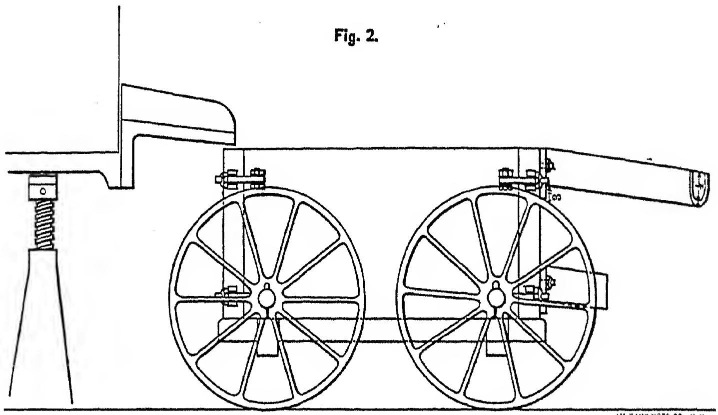
that the top of the pot, when in position, shall be on a level with the dump at this point. Those pits with their large pots, of which there are several, are located in a row about 120 feet in front of the blast-furnaces. When one of these large pots is full of slag and matte, it is hoisted and carried by an overhead crane to such a position that when the plug of clay is punched out the slag will run into side-dumping slag-pots on trucks, after which the large pot is conveyed to another position, so that the matte left in it may be tapped into a pot of the same construction, but somewhat larger than the ordinary slag-pot. The shell in this large pot, after the slag and matte have been tapped, is conveyed to a convenient spot, and the "crusts" are dumped.

When the side-dumping slag-pots have been filled, they are pulled by mules to the edge of the dump, and the slag is thrown over. The matte is allowed to cool in the pot into which it was tapped, and is then broken with culling-hammers. Any slag on the dump, in the form of crusts or otherwise, which is to be re-smelted, is loaded into a small car lowered on a tramway, so that its top is below the level of the dump, and hoisted in this to a tramway several feet above the charge-floor, upon which the car is wheeled to the furnace at which it is desired to dump the slag. The idea of keeping the slag and matte hot for a reasonable length of time, so as to allow the particles of matte to separate from the slag, seems good, and is effected by placing the pot containing a comparatively large amount of molten material in the pit mentioned above, thus allowing a minimum amount of cold air to come into contact with the sides of the pot. The tapping of the slag and the matte from this pot is simply an application of the Devereux system.

The Arkansas Valley Smelting-Works.—These works, situated at Leadville, Colorado, are a branch of the Consolidated Kansas City Smelting and Refining Company, of Argentine, Kansas. Here slag is handled from seven lead blast-furnaces and three copper-matte blast-furnaces. The establishment is comparatively old, having been smelting ore for more than sixteen years, and the slag-dump extends several hundred feet from the furnaces. The installation of arrangements for granulation would be expensive, and the method used for handling slag seems the best under existing conditions. In front of the furn-

ace building, about 20 feet, and for its full length, there is an open cut about 5 feet deep by 6 feet wide, into which is backed

Fig. 2.

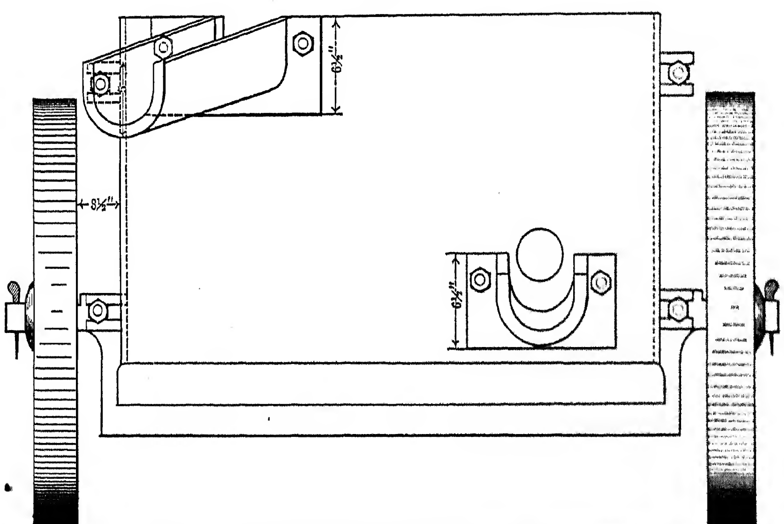


Scale ⅛ inch = 1 foot.

AM. BANK NOTE CO., N.Y.

Furnace-front and Fore-hearth, Furnace No. 1, Arkansas Valley Works. Side view.

Fig. 3.



Scale 1 inch = 1 foot.

AM. BANK NOTE CO., N.Y.

Fore-hearth, Furnace No. 1, Arkansas Valley Works. Front view.

a train of trucks carrying two Nesmith side-dumping slag-pots each. (See illustrations in Hofman's *Metallurgy of Lead*, Figs. 140 to 144, page 204.)

Furnaces Nos. 1, 2, and 3, are matting-furnaces, located a one end of the row of ten furnaces; and as No. 1 differs in details of construction from Nos. 2 and 3, I will speak of them separately.

No. 1 is a matting-furnace, the bottom-plate of which is elevated on jack-screws 34 inches from the floor of the furnace room. The product from this furnace, as from the other two matting-furnaces, is a copper-iron matte, running from 12 to 35 per cent. in copper.

Fig. 2 gives a side-view of the front of the furnace and the fore-hearth, and Fig. 3 the front-view of the fore-hearth. The molten material from the furnace is tapped periodically into this fore-hearth; the matte settles to the bottom and is tapped into a "shell" (see Figs. 4, 5, and 6), and the slag overflows into a settling-pot (see Fig. 1). Any matte that has not settled from the slag in the fore-hearth has another chance to settle in this pot. The matte is drawn from the lower tap-hole of this settling-pot into a "shell," and the slag overflows into an ordinary slag-pot. This latter is wheeled to the edge of the cut in the dump mentioned above, and by tilting the pot, dumped into one of the slag-pots in the open cut. The shell of slag that adheres to the sides of the pot after emptying the bulk of it into the pots in the cut, is removed from the pot and hoisted in a car to the furnace-floor to be remelted, in much the same manner as slag to be resmelted is treated at the Omaha and Grant smelting-works at Denver, Colo., as described already.

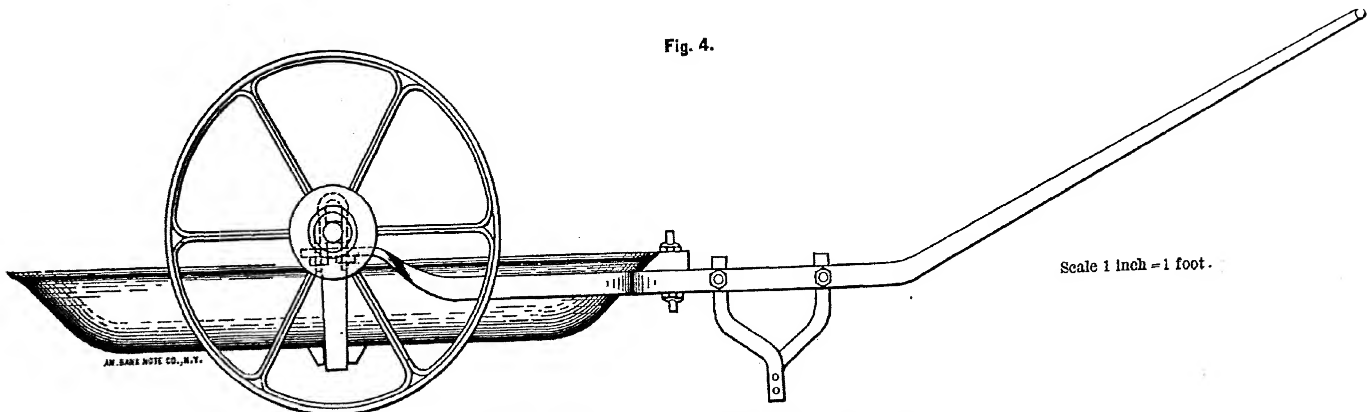
The matte in the "shells" is dumped near the railroad-tracks, conveniently for loading for shipment.

The fore-hearth of this furnace, owing to the irregularity of the charges, is changed about twice a month; the large settling-pots are changed three or four times a day.

On furnaces No. 2 and No. 3 a different style of fore-hearth is used. Fig. 7 shows the "side-elevation," and Fig. 8 the front plate of this fore-hearth. In this case the molten material runs into the fore-hearth near its bottom, and the matte is drawn from the lower tap-hole into an ordinary slag-pot. The slag rises and overflows into a slag-pot; from this point on both slag and matte are treated like those of furnace No. 1.

So far as obtaining clean slag is concerned, this latter system of fore-hearth works more satisfactorily than the style used on

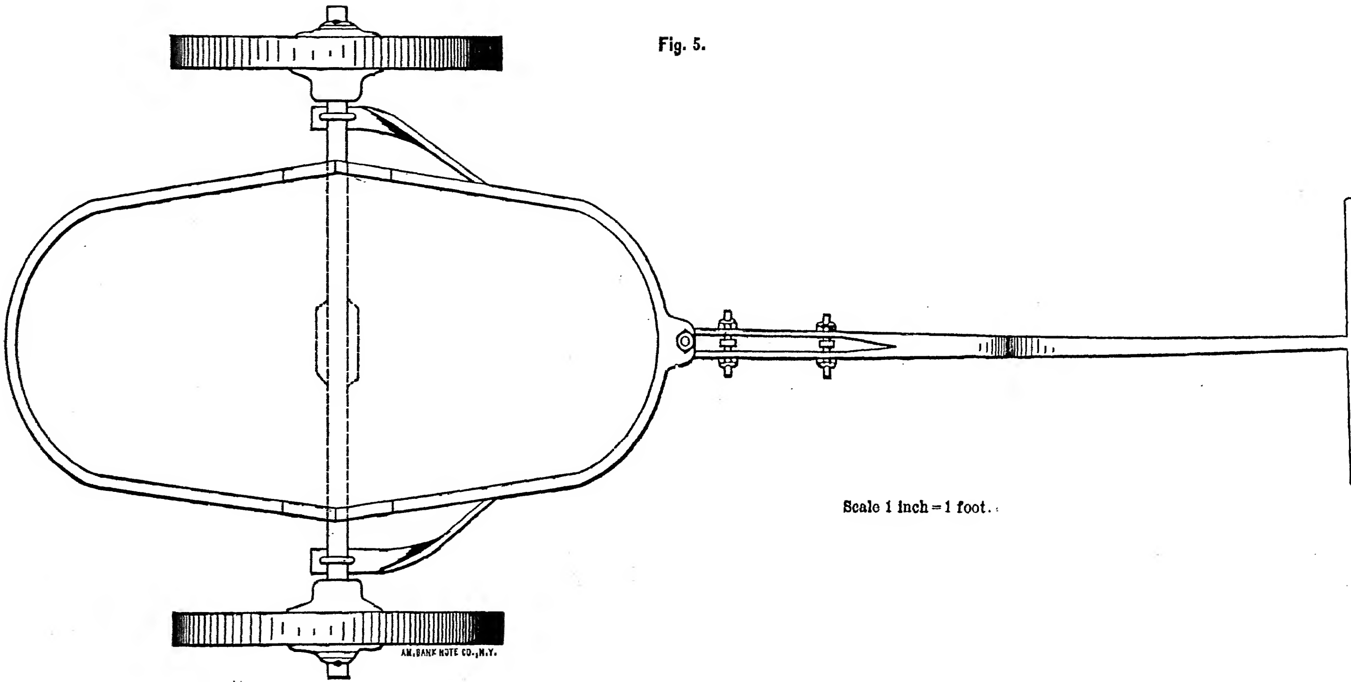
Fig. 4.



Scale 1 inch = 1 foot.

Shell for Matte, Arkansas Valley Works. Side view.

Fig. 5.



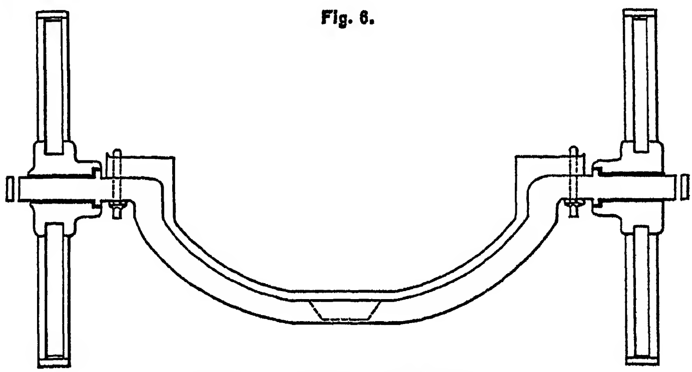
Scale 1 inch = 1 foot.

Plan of Fig. 4.

furnace No. 1, and does away with the use of the settling-pot. The design was recommended by Dr. E. D. Peters, Jr.

At the lead-furnaces the molten material is run into Devereux pots which are wheeled on to grates, covering in front of the lead-furnaces the cut into which are run the slag-trucks. The plug of clay in the Devereux pot is punched out, allowing the slag to flow through an opening in the grates (18 by 6 inches), into one of the pots in the cut. Then, this Devereux pot, containing lead-matte, and the small amount of slag left in the pot below the hole, is wheeled right on over the grates to the culling-ground beyond, and, when cool enough, its contents are

Fig. 6.



Vertical Cross-section of Fig. 4.

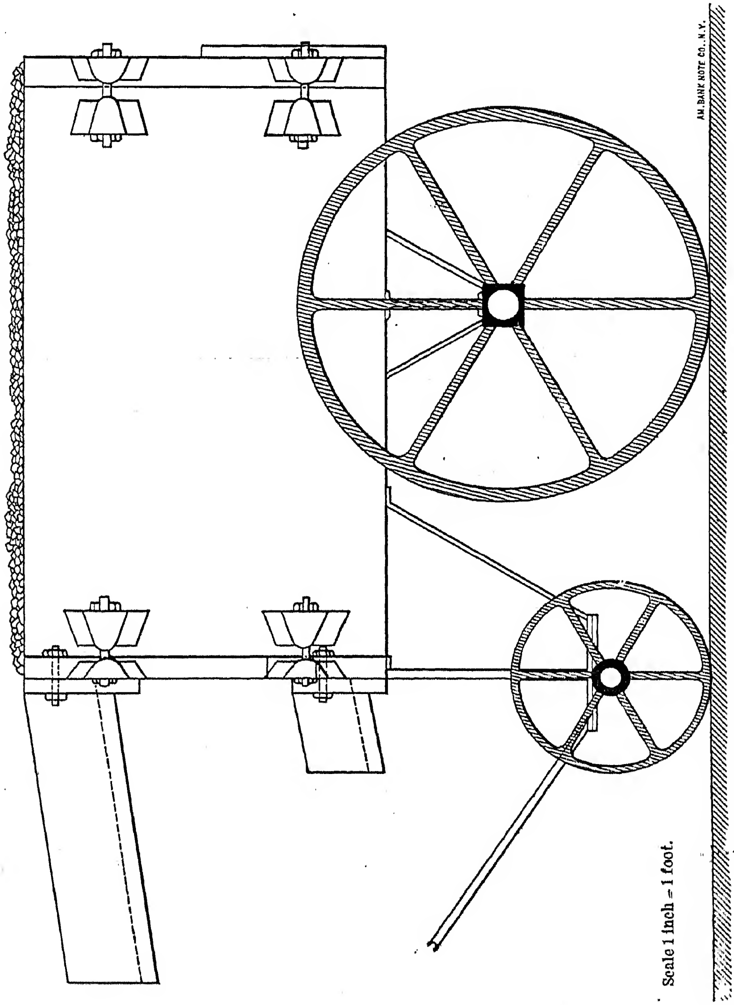
dumped. The matte and slag are then separated, the matte being loaded into Thatcher dumping-cars, conveyed to the crusher, roasted and slagged, and finally re-treated in the matting-furnaces.

The slag, if "dirty," is conveyed to the furnace-floor, and re-treated; if not, it is thrown over the dump.

The matte made in the lead-furnaces at a large custom smelter of this kind, necessarily varies in amount; so the hole in the Devereux pots used is made high up on the front side of the pot. In front of each opening in the grating over the cut is a small wrought-iron step-like arrangement, by the use of which under the "rest" attached to the handle of the pot, the flow of slag may be regulated, according to the amount of matte being made. If a very small amount, the "rest" of the slag-pot is placed on the highest step.

When all the pots of a train in the cut are filled with slag, a small locomotive conveys them 400 feet or more to the edge of

Fig. 7.



Fore-hearth, Furnaces Nos. 2 and 3, Arkansas Valley Works. Side elevation.

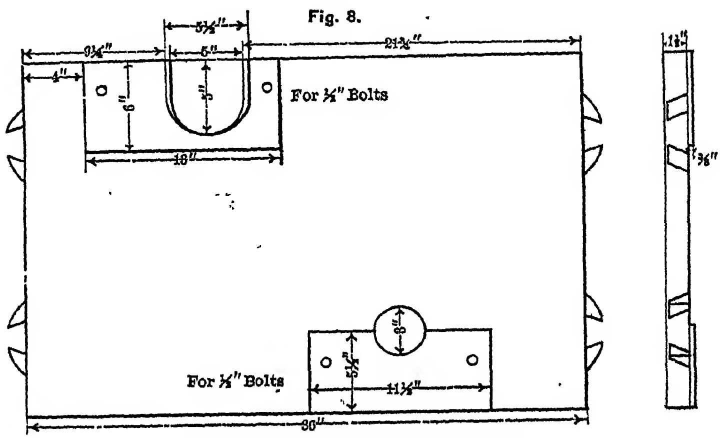
the dump, where the slag is thrown over. The locomotive immediately places another train of empty pots in the cut.

Since the above was written concerning the system of slag-manipulation at the Arkansas Valley works, Mr. R. D. Rhodes has constructed there a reverberatory-hearth, into which the

slag and matte are run from the blast-furnaces; and by keeping up a slight heat in the fire-place of this hearth, a separation of slag and matte is said to be economically effected.

The American Smelting Works.—At these works, in Leadville, Colo., a branch plant of the Aurora Smelting and Refining Company, the slag from seven lead-furnaces is granulated, as is likewise most of the matte made. The molten material is tapped from the furnace into a settling-pot. A portion of the matte settles in this pot, and that which does not, overflows with the slag into a second, somewhat smaller, settling-pot. From this the slag overflows into a jet of water, under 20 pounds pressure,

Fig. 8.



Front Plate of Fore-hearth in Fig. 7.

n which it is granulated and sluiced away. When the second settling-pot is full of matte, it is replaced with an empty one, and hoisted to a suitable point, and the matte is granulated, the granular particles falling into storage-bins, from the bottom of which the stock is drawn out, when desired, into tram-cars and conveyed to the roasters.

The Consolidated Kansas City Smelting and Refining Co.—The main works of this company are at Argentine, Kansas. Here slag is handled in somewhat the same way as at the works in Leadville, with the exception that the Argentine slag, instead of being thrown away, is sold to the railroads as ballast. The molten material is run from the furnace into Devereux pots, which are wheeled to a cut somewhat similar to that described

at the Arkansas Valley Works. Into this cut are run slag-trucks, which, however, carry, instead of conical pots, rectangular cast-iron boxes. When a train of these boxes is ready for the slag to be tapped into them, a wrought-iron pronged eye-bolt 1½ inches in diameter is placed in the middle of each box, the eye extending about 4 inches above the top of the box. When all the boxes on the trucks have been filled with slag, a small motor side-tracks the train and puts another train of empty boxes in the cut. After the slag in the boxes has cooled sufficiently, a small travelling crane picks up one cake of slag after another by means of a hook inserted into the eye of the pronged eye-bolt, and deposits the same in front of a crusher. The cakes are broken up with hammers, the eye-bolts are withdrawn, and the slag goes through the crusher, and is loaded on railroad cars, ready for use as ballast.

The Omaha and Grant Smelting Company.—At the works of this company in Omaha, another adaptation of the Devereux system is in use. The molten material is tapped into the Devereux pots from the blast-furnaces and allowed to stand long enough for as complete a separation of slag and matte as possible. About 75 feet in front of the furnace-building, and centrally located, is a contrivance which handles, very ingeniously, the slag as tapped from the Devereux pots. It consists of a circular pit, about 35 feet in diameter, and filled with water, upon the outer zone of the surface of which an annular disk of cast-iron plates, about 2 feet wide and 1¾ inches thick, and slightly trough-shaped on the top, is made to revolve. This disk is supported partly by radial arms from a pillar in the center of the pit, and partly by rollers attached to the plates underneath, and travelling on stationary plates which are in turn supported by pillars around the circumference of the pit. Revolution is effected by means of a rack on the periphery of the disk. A constant stream of cold water is run into the pit, and an outlet is provided for the heated water, the supply being so regulated that the water shall not overflow the trough-shaped surface of the disk, but the lower side of the plates, with the rest of the mechanism, shall be constantly immersed in cold water. The apparatus makes one revolution in five minutes.

A Devereux slag-pot having been wheeled to the edge of the

pit, the slag is tapped, first striking a cast-iron plate fixed at an angle of about 60 degrees from the horizontal, to prevent the undue scattering of the slag, which passes through upon the the revolving trough-shaped disk. On this it spreads out, covering the plates to the depth of ¼ to 1 inch, according to the amount of slag being made. From the molten state in which it was tapped from the pot, the slag cools gradually on those water-cooled plates, until at the end of a revolution it is quite brittle. A piece of ½-inch steel is set diagonally across the revolving plates in such a manner as to break up the slag, and force it on to a conveyor, which takes it up into bins, from which it is loaded on railroad cars. It is of such size and structure as to be of superior quality for the purpose of ballast, being square or rectangular in shape, from ¼ to 1 inch in thickness, and from ½ inch to 4 inches in surface area. This arrangement was designed by Mr. Page, the superintendent at these works.

The Hanauer Smelting Works.—At these works, near Salt Lake City, Utah, silver-lead ores are concentrated in blast-furnaces. A sluice carrying water under pressure is run under the floor in front of the furnaces. The slag and matte are tapped periodically from the furnace into settling-pots; the slag overflowing into the sluice is granulated by the stream of water, and carried to a pit or "boot," whence it is hoisted by a belt-elevator into bins, from which it is loaded on railroad-cars to be used for ballast. The matte and the small amount of adhering slag in the settling-pots are culled and handled in the usual manner.

Slag-Tiles.—As a matter of particular interest, I will briefly describe, in conclusion, a successful manufacture of slag-tile from blast-furnace slag, invented by Mr. Marco Chiapponi, E.M., which I saw in operation at the smelting-works of Señor Concha y Toro, about 15 kilometers from Santiago de Chili.

The slag and matte are tapped from the blast-furnace into a slag-pot, and, after settling a few moments, the slag is poured from ladles into moulds (6 by 1 inch each); these are placed on a hearth which has a movable cover; the moulds being filled with slag, a cover is placed on them, as well as on the hearth. A very light heat is kept up, so that the slag is very slowly cooled; and when it appears black, the moulds are lifted

from the hearth and the slag-tiles are dumped into cold water. The slag-tiles thus made are light and portable, and, when laid, tough and durable. A slag carrying a considerable excess of iron has been preferred. These tiles are sold for from $30 to $60 (pesos Chilenos) per 1000.

It is generally known that it is impossible under ordinary circumstances to make tiles of such dimensions as are mentioned above, without some extraneous means for toughening.

Though, at other establishments than those spoken of, I have observed somewhat altered details in the systems of slag- and matte-manipulations, I believe this paper covers essentially my purpose, which was to describe the main features of our Western practice.

The Volatilization of Silver in Chloridizing-Roasting.

BY L. D. GODSHALL, EVERETT, WASHINGTON.

(Pittsburgh Meeting, February, 1896.)

THE latest revised edition of Mr. C. A. Stetefeldt's book on the Lixiviation of Silver-Ores, which appeared very recently, contains no mention of the volatilization of silver in chloridizing-roasting—an omission which is the more remarkable in view of the fact that in former editions of the work this important subject was noticed. Moreover, Mr. Stetefeldt has discussed it to some extent in his paper on "The Stetefeldt Furnace," * in which he criticised certain statements made by me before the Colorado Scientific Society in May, 1893, in a paper entitled "A Review of the Russell Process;" † and Mr. Morse, in his paper on "The Lixiviation of Silver-Ores by the Russell Process at Aspen, Colorado," ‡ has given statistics showing the loss of silver by volatilization (including the dust-loss, which Mr. Morse estimates at not more than 1 per cent.) on more than 30,000 tons of ore roasted, to have been 9.16 per cent. The matter must certainly be deemed important enough to warrant me in returning to it, and offering, with the aid of the figures made public since my first paper, a reply to the criticisms of Mr. Stetefeldt.

The passages of my paper which he deemed to be "ques-

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* *Trans.*, xxiv., 3. † *Proc.* Col. Sci. Soc.

‡ Florida Meeting, March, 1895, *Trans.*, xxv., 137.

tionable conclusions from a limited experience with the Stetefeldt furnace at the Holden mill, Aspen, Colorado," were quoted by him,* and are here repeated for the convenience of the reader:

"The use of this furnace has certain limits beyond which it is unwise to go. Siliceous ores carrying 6 per cent. or more sulphur can be chloridized better in some other furnace. Ores carrying a large percentage of lime—say from 15 to 20 per cent. of CaO—are also very difficult to chloridize properly in the Stetefeldt furnace, unless sufficient sulphur is present to combine with the CaO and form $CaSO_4$; even then the chloridization in the furnace is frequently very low, and rarely exceeds 60 per cent.

"The writer has seen the furnace deliver roasted ore from certain mixtures high in lime and sulphur where not more than 15 to 20 per cent. of the silver had been chloridized in the shaft of the furnace, where about 65 per cent. of the ore is roasted. However, after the above ore had been lying on the cooling-floor for three days, fully 90 per cent. of the silver was found to be chloridized. The extremely low percentage of the silver converted into chloride in the furnace was due partly to its having been crowded beyond its capacity, but chiefly to the large percentage of sulphur present causing such a strong reducing atmosphere of sulphur dioxide that the effect of the chlorine liberated by the acid gases was neutralized. Frequently the odor of sulphurous acid escaping from the roasted ore as it was discharged from the shaft of the furnace was sufficiently strong to overcome all smell of chloride fumes. The sulphur in the raw charge of the ore just considered ranged from 8 to 10 per cent., being about half that of the lime and magnesia present."

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"Can such ores be roasted better with any other furnace? Metallurgically, the answer is, Yes. The proof of the above assertion with regard to Aspen ores has been demonstrated by the writer by roasting in a reverberatory furnace 10 lots of ore containing 25 per cent. of CaO (MgO not determined, but probably amounting to 10 or 12 per cent.) with less than 2 per cent. of sulphur, and using, practically, the same amount of salt as in the Stetefeldt furnace. The chloridization was all that could be desired."

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"For ores containing from 3 to 8 per cent. of sulphur, the Brückner, Pearce or Howell-White are to be recommended."

To these statements Mr. Stetefeldt makes two general objections: first, that they were based on too limited an experience; second, that they were not accompanied with adequate data in the way of proof. As to the first, I venture to submit that while my experience was confessedly limited to the ores of a certain district, it comprised nearly a year of continuous observation, practice and experiment, and was, therefore, quite long enough to warrant the formation of an opinion concerning the materials to which it referred, and, by analogy, other similar materials. As to the second objection, I have only to

* *Trans.*, xxiv., 10.

say that, at the time I wrote the Aspen works were still in operation, and the company was unwilling to have exact details made public. This deficiency has been completely made up through the later publication of complete statistics by Mr. Morse in his paper, above cited.

Mr. Stetefeldt points out what seems to him an inconsistency between my statement that "siliceous ores carrying 6 per cent. or more sulphur can be chloridized better in some other furnace" than the Stetefeldt, and my subsequent statement that "for ores containing 3 to 8 per cent. of sulphur, the Brückner, Pearce, or Howell-White are to be recommended." But he will perceive, on reflection, that the two propositions are not inconsistent. The hypothesis, for instance, that ores carrying from 3 to 6 per cent. of sulphur could be chloridized as thoroughly by the Stetefeldt as by either of the three rival furnaces named, but that, for such ores, one of the three was to be recommended on other grounds, such as smaller cost of construction, or smaller loss in roasting, would remove the fancied contradiction.

Leaving these preliminary and subordinate criticisms, let us consider the first important point urged in opposition to my conclusions, namely, that the Aspen ore is exceptional and peculiar, and must be acknowledged to be "a difficult ore to chloridize in the Stetefeldt as well as in any other furnace." This difficulty in a Stetefeldt furnace is admitted by all who have made the attempt to overcome it; but the proposition cannot be admitted as regards "any other furnace."

On page 103 of Mr. Stetefeldt's book is given a description of the manner in which an ore should be tested to determine its fitness for lixiviation. It is there observed, that roasting-tests are best carried out in the muffle of an assayer's cupelling-furnace, in clay dishes about 4½ to 5 inches in diameter, holding a charge of 3½ A. T. On the following page we read:

"It is not always possible to produce by muffle-roasting, on a small scale, the same effect that can be obtained by actual mill-work; especially if an ore is treated requiring banking up on the cooling-floor for many hours, in order to reach a high chlorination of the silver. Hence, an unfavorable result is not always a proof that the ore is unsuitable for lixiviation."

The natural conclusion is, that if an ore is easily chloridized in small quantity, in an assay-furnace, it would be as well, or better, chloridized on a commercial scale in a large furnace. As to the correctness of this proposition, I will say nothing

here; but, accepting it for the present as true, I give the results of its application to the Aspen ores.

The experiments, the results of which are recorded below, were made as follows: The raw ore was crushed so as to pass through an 80-mesh sieve, 150 grammes taken and thoroughly mixed with 16 per cent. salt and 4 per cent. of iron pyrites containing about 40 per cent. of sulphur, roasted from 15 to 30 minutes, commencing with a very low and ending with a light-red heat. After cooling, the roasted mixture was carefully removed from the dish and weighed, and again assayed to determine the loss by volatilization, the percentage of soluble salts also being determined. Leaching-tests were now made to determine the percentage of silver-chloride formed in the roasting, and the proper method of applying the Russell-solution to obtain the highest possible extraction of the silver.

TABLE I.—*Laboratory Experiments in Chloridizing Aspen Ores.*

Name of Ore.	Ounces Ag per ton.	Analysis.						Silver Chloridized.	Best Extraction by Russell Process.
		SiO_2.	Fe.	CaO.	MgO.	$BaSO_4$	Zn.	Per ct.	Per ct.
Justice	13.2	14.9	20.8	6.5	.....	33.1	3.5	96.2	98.9
Sylvanite	36.0	53.8	11.1	5.4	.....	12.4	.....	92.0	96.5
Camp Bird	20.9	43.6	7.3	12.0	.....	22.4	0.8	98.6	99.0
Pride of Aspen	44.5	27.4	3.1	27.6	8.0	0.5	1.4	95.6	98.5
Compromise	12.5	44.2	4.3	17.1	.....	.....	0.8	95.9	97.0
Percy	25.5	18.1	8.2	18.9	.....	21.8	2.0	98.0	99.6
Franklin	41.8	28.6	4.3	21.8	.....	9.0	2.1	96.7	98.6
Smuggler	64.2	2.8	2.2	6.5	.....	78.0	2.4	90.9	97.9
Aspen	19.9	15.5	2.6	25.5	.....	.....	.....	96.5	98.2
Mixtures of Different Samples of Aspen Ores.									
No. 1–2 Samples	46.0	33.1	4.0	4.6	0.7	46.4	2.0	98.3	95.7
" 2–2 "	32.2	6.8	2.2	16.6	.....	41.5	3.3	95.1	96.3
" 3–3 "	38.0	25.2	3.4	12.5	2.8	33.2	2.2	98.5	98.5
" 4–9 "	23.2	18.3	8.5	20.2	.....	18.7	2.0	97.9	99.2
" 5–19 " *	25.5	21.8	8.5	19.2	3.4	13.4	3.1	97.7	98.5
" 2	32.2	Roasted with 4 per cent. pyrite. 12 " salt.						91.4	97.3
" 4	23.2	" " "						97.0	.....
" 4	23.2	" " 4 " pyrite. 10 " salt.						91.9	96.6
" 4	23.2	" " 4 " pyrite. 8 " salt.						94.9	.....

* Attention is called to these results, which were obtained without the addition of any pyrite in the chloridizing-roasting.

A great many experiments with results similar to the above might be recorded here. In fact, the results were all so uniformly good, that they soon became monotonous, and the experiments were discontinued.

In view of the above results, I claim that Mr. Stetefeldt is not warranted in making the statement that the Aspen ores are difficult to chloridize in any furnace. Mr. Stetefeldt says that the fact that the sulphur in the Aspen ore (as mixed for roasting) exceeds the limit given by me from 2 to 7 per cent., would force one to the conclusion that the Stetefeldt furnace at the Holden mill was a complete failure, and that Mr. Morse's statistics contradict this. Instead of contradicting, Mr. Morse has since confirmed this fact very conclusively.

Mr. Stetefeldt says:

"It is well known that high chlorinations of silver-ores containing a large percentage of pyritic minerals can only be obtained after roasting in a Stetefeldt furnace by leaving the discharged ore for twenty-four hours, or longer, in heaps on the cooling-floor. But why should this be made an objection?"

In my review of the Russell process, reasons were given for considering heap-roasting on the cooling-floor extremely objectionable. Mr. Morse, in his paper on "The Effect of Washing with Water upon the Silver Chloride in Roasted Ore,"* while differing with me as to the reactions involved, is equally emphatic in condemning this heap-roasting for ores sustaining as much zinc (from 1 to 3 per cent.) as those treated at Aspen. And Mr. Stetefeldt himself, on page 65 of the last edition of his book, accepting Mr. Morse's theory as to the cause of the trouble, indirectly condemns heap-roasting and the Stetefeldt furnace for all ores containing from 1 to 3 per cent. of zinc, and a sufficient amount of pyritic minerals to make such supplementary heap-roasting necessary.

He controverts my statement that the supply of free oxygen is extremely limited in the heap on the cooling-floor. The reactions given by him as occurring in this heap-roasting are undoubtedly correct, but he does not show that these are the only ones which occur; and in any event, the fact that three or four days are required to oxidize by means of these reactions the comparatively small amount of sulphur still in the ore, seems to me conclusive evidence that the admission of air to the in-

* Atlanta Meeting, October, 1895, *Trans.*, xxv., 587.

terior is very slow and limited. This impression is confirmed by the fact that the crust of the heap chloridizes in as many hours as days are required to chloridize the interior to an equal degree.

My theory as to the loss of silver by volatilization in chloridizing-roasting is declared to present nothing new. But the question of originality is not so important as the question of fact.

In the first edition of his book, Mr. Stetefeldt said:

"It is a well-established fact that the loss of silver by volatilization in roasting in a Stetefeldt furnace is a minimum and almost imperceptible, this loss being principally a function of time."

It was this statement which I controverted in the paper which he criticizes. What he now says is:

"The evaporation or volatilization of all substances is governed by the same general laws. The effective elements are: time, temperature, surface exposed, character of the atmosphere in which evaporation or volatilization takes place, density or pressure of the latter, and its motion or exchange in relation to the substance evaporated or volatilized. Thus, for instance, more silver is volatilized in roasting a small ore sample in a muffle than in actual reverberatory-furnace work, *because more surface is exposed, and the particles have more contact with air in the former case.*" (The italics are mine.)

In the paper criticized, I said on this point:

"Mr. Stetefeldt claims that the volatilization of silver is principally a function of time. If the above statement is true, it is one of the strongest arguments in favor of the Stetefeldt furnace. Experiments made to determine this point proved that time was an important function under certain conditions, but not at all under all conditions. There is another factor which, so far as I know, has never been mentioned in connection with the loss of silver in chloridizing-roasting, which factor is air, or oxygen. That oxygen is one of the principal if not the principal cause of the volatilization of silver in chloridizing-roasting, is believed by the writer for the following reasons:

"It is possible to volatilize from 40 to 60 per cent. of the silver contents in an ore in a chloridizing roast conducted in a muffle at a low, red heat, in fifteen to thirty minutes. The same ore roasted at the same temperature with the same percentage of salt on a commercial scale, in a large furnace, under the worst conditions, would not show more than one-third of the loss sustained in the muffle, while under the best conditions the loss would, in all probability, not exceed one-tenth of that experienced in the muffle. The time required for roasting in any large furnace other than the Stetefeldt would be many times that required for the muffle roast. If it can be shown that, with only a momentary exposure of the ore, as is the case in the Stetefeldt furnace, the percentage of silver volatilized is as high, or nearly as high, as when the ore is roasted for eight hours with the same quantity of salt in a reverberatory furnace, it must be concluded that there are other influences more important than time governing such loss. This has been

found to be the case with the Aspen ores. The average loss by volatilization determined in roasting some 20-ton lots of ore in a reverberatory was found to be less than that experienced at Aspen, where the Stetefeldt furnace is used.

"In the above cases the following conditions were the same: Character of ore, percentage of salt used and temperature of roasting. Those not the same were: Time of roasting and amount of oxygen in contact with the particles of ore during the time the chloridizing action was going on.

"Time only increases the volatilization of silver when sufficient heat and air are present. The highest loss was sustained in the muffle where the amount of air used in roasting was greatest. The ore roasted in the reverberatory was just as well chloridized and gave fully as high extraction as that treated in the Stetefeldt furnace.

"It was also noticed at Aspen that some of the lowest losses in silver were experienced during the months when a heavy excess of sulphur had been used and only a limited supply of air allowed to enter the furnace; it was also afterwards observed that additional air produced a higher chloridization of the silver, but that the losses by volatilization were also higher. Every indication pointed to the fact that the higher chloridization in the furnace was obtained at the expense of part of the silver. The fact of a smaller loss by volatilization when roasting with a higher percentage of salt can only be explained on the same theory, namely, that the atmosphere enveloping the ore is one of chlorine rather than oxygen. During the strong chloridizing action on the cooling floor where the ore remains red-hot to within a few inches of the surface for two, and frequently three days, the loss by volatilization is not perceptible, and samples taken from different parts of the heap at different intervals show no variation in value. It is true that all fumes are condensed and prevented from escaping into the air by the cold crust of ore, and the volatilization might be prevented more by a mechanical than chemical condition.

"In chloridizing-roasting gold-ores, the theory equally holds so far as the agency of air is concerned. The writer has roasted gold-ores side by side in Brückner and Howell-White furnaces with the same percentage of salt in each. One of the Howell-White furnaces was built in front of the other and the ores allowed to pass through both. Nearly all the sulphur was driven off in the first furnace. The salt was added to the hot ore as it was being fed from the first furnace into the second, where it was roasted for a little over an hour. In the Brückner the salt was added to the raw ore and the charges roasted from eight to twelve hours. The volatilization of gold was the heaviest in the Howell-White furnace, where undoubtedly each particle of ore was more frequently exposed and longer in contact with the air than in the Brückner furnace. The same ore, when roasted in the muffle with the same percentage of salt at a low red heat for fifteen to thirty minutes showed nearly ten times as great a loss in gold by volatilization as was experienced in the practical operation.

"It is evident from the above statements that determinations of the volatilization of silver and gold by chloridizing a few hundred grammes of ore in a muffle are not apt to be very reliable. According to Mr. Letts, Yedras ore is now being roasted (chloridized) in the reverberatory furnace at a loss of only 6½ to 7 per cent. of the silver."

To meet the objection that the above argument was unaccompanied with detailed proof, I offer the following, which I trust will be convincing:

TABLE II.—*Results of Aspen Ore Roasted in Muffle Furnace.*

Name of Ore.	Amount Taken.	Salt.	Sulphur.	Time of Roasting.	Silver Volatilized.
		Per cent.	Per cent.		Per cent.
Smuggler		16	2		35.7
"		16	2		70.6
Percy	From 50	16	2	From 15	53.5
"		16	2		46.5
Compromise		16	2		45.5
Mixture No. 1-2 Samples	to 150	16	2	to 30	51.9
" 2-2 "		16	2		46.1
" 3-3 "	grammes.	16	2	minutes.	33.4
" 2-9 "		16	2		30.6
" 5-19 "		16	2		29.4

TABLE III.—*Results of Aspen Ore Roasted in Stetefeldt Furnace.*

Ore.	No. of Tons.	Salt.	Sulphur.	Silver Volatilized.	AgCl formed in Furnace.	Decrease of AgCl by Washing.
		Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
February run	2090	11.32	9.49	7.4	37.04	32.41
March "	1319	11.52	8.18	7.62	43.32	37.16
April "	2435	9.75	8.48	4.9	46.94	19.58
July "	2123	12.03	7.3	10.8	53.51	8.6
August "	2687	11.61	7.1	11.18	59.82	3.32
September "	2656	11.27	7.4	11.57	56.78	0.51

TABLE IV.—*Results of Aspen Ore Roasted in Reverberatory Furnace.*

Name of Ore.	Amount.	Salt.	Sulphur.	Time of Roasting.	Silver Volatilized.
		Per cent.	Per cent.		Per cent.
Aspen		16	2		5.7
Camp Bird	From 1 to	15	2	From 6	0.8
Durant		10	2		4.8
Percy	2 tons of	10	2	to 9	5.4
Justice		15	2		4.3
Compromise	each.	10	2	hours.	0.7
J. C. Johnson		15	2		7.9

The above results need no comment, so far as the comparative volatilization of silver is concerned. Mr. Stetefeldt argues that the loss at Aspen might have been in dust escaping from the last dust-chamber to the chimney, and hence not fairly chargeable to volatilization. He cites the experience at the Ontario mill, where in flues about 150 feet long, 4 feet wide

and 6 feet high, interposed between the dust-chambers and the chimney, 1.26 per cent. of the silver in the ore is recovered. It would be interesting to know the composition of the material saved in these flues, and especially the percentage of condensed fume containing silver previously volatilized. At Aspen, a considerable amount of such material was saved in the rear end of the dust-chamber, as may be seen from the following table:

TABLE V.—*Chloridized Ore-Samples, Beginning with No. 1 Taken from Shaft of Stetefeldt Furnace, and Ending with No. 21 Taken from Foot of Stack.*

No.	Ounces Ag per ton.	Pb.	Solubility.	No.	Ounces Ag per ton.	Pb.	Solubility.
		Per cent.	Per cent.			Per cent.	Per cent.
1	30.0	2.6	16.5	12	18.2	6.6	32.5
2	26.8	2.0	10.3	13	18.2	7.2	33.5
3	21.4	0.6	7.7	14	19.0	7.9	35.7
4	21.3	Tr.	6.6	15	19.2	9.9	38.2
5	17.0	1.0	11.7	16	19.7	13.2	38.5
6	17.0	2.3	15.7	17	20.3	17.8	40.2
7	18.4	3.0	18.8	18	24.0	14.0	30.5
8	17.2	3.0	20.3	19	24.4	10.5	26.4
9	18.8	3.9	22.8	20	21.6	13.2	40.5
10	19.1	4.6	24.0	21	22.2	12.9	36.2
11	17.5	5.0	26.0				

The actual volatilization of silver in the furnace itself, at Aspen was therefore even greater than the figures I have previously given, by the amount of volatilized silver thus recovered.

That the conditions above stated are not peculiar to Aspen ores exclusively, was indicated by the treatment of several hundred tons of Creede ores in Aspen furnaces. These ores contained approximately 90 per cent. of silica, no lime, and 1 per cent., or less, of sulphur. The detailed results are not in my possession; but I am informed that the chloridization in the furnace was remarkably good, but that the loss by volatilization was even greater than on any of the Aspen ores.

Mr. Stetefeldt concludes his paper with the frank admission that "the fact that the loss of silver by volatilization during chloridizing-roasting in the Stetefeldt furnace is a minimum, under all circumstances, as compared with roasting in any other furnace, lacks absolute proof." In view of the foregoing

discussion, I think we may fairly ask for evidence that such loss in that furnace is a minimum under any circumstances.

Postscript.—Since this paper was put in type, the writer has received the news of the sudden death of Mr. Stetefeldt. If he had known earlier of this event, while, of course, it would not have altered the opinions and arguments stated by him, it would certainly have led him to express his cordial sense of the professional ability of Mr. Stetefeldt, the value of his work, and the loss inflicted upon the Institute by his death.

The Hydraulic Elevator at the Chestatee Mine, Georgia.

BY W. R. CRANDALL, DAHLONEGA, GA.

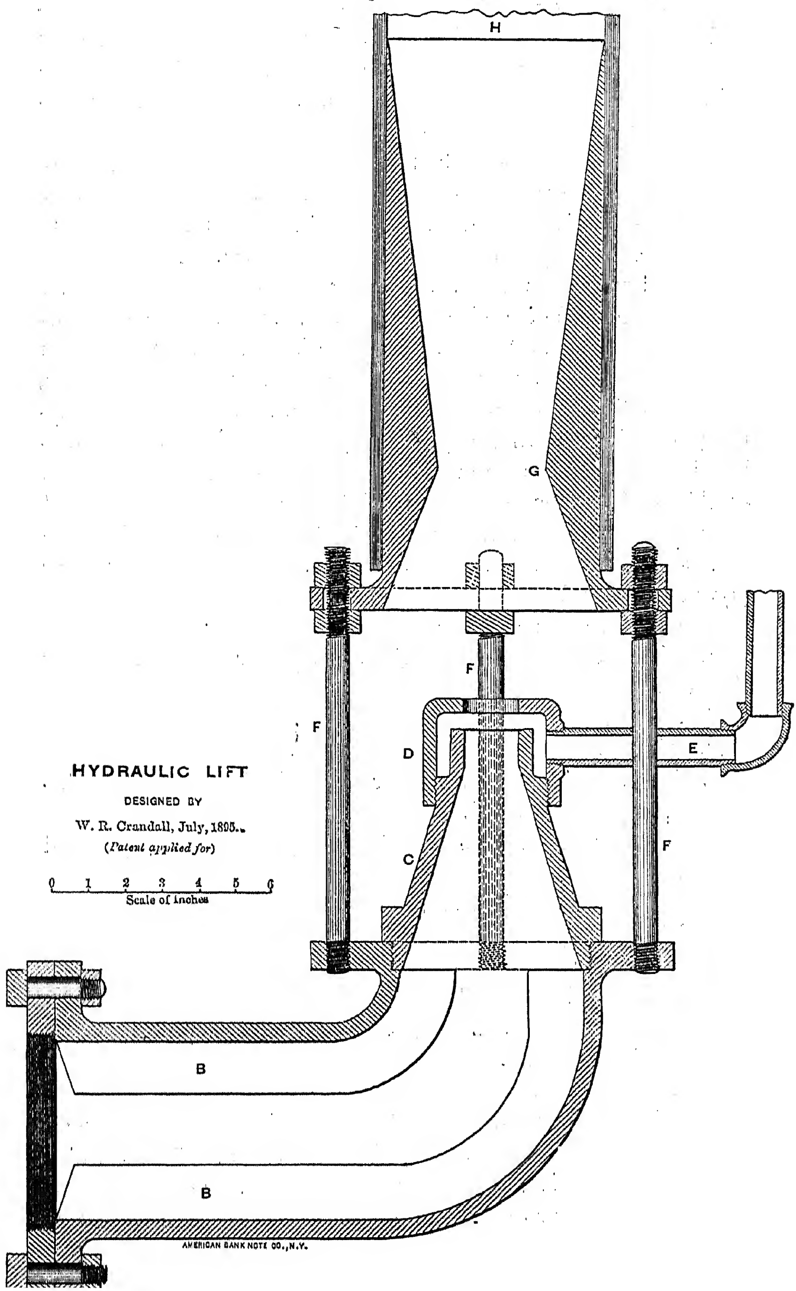
(Pittsburgh Meeting, February, 1896.)

The southern gold-fields offer some of the most complex and trying problems encountered in mining; and their successful solution often means the success or failure of the particular enterprise involved.

In vein-mining, where the values are scattered over large areas, making it necessary to handle a large tonnage of very low average grade, the successful solution has been found in some instances by a combined hydraulicking and milling; water under pressure being used as the mining agent, and also for transporting the broken material to the mills, and effecting in its passage a partial concentration of the product by washing out the lighter and looser dirt of the ore-mass.

The placer-deposits of the South, which were the original localities from which gold was mined, have been extensively worked, and to any one now undertaking their development offer many difficulties. The antiquity of the industry in this field, which was the cradle (in fact, literally the *rocker*), in which the California '49-er was born, gives assurance that all the deposits available to water and natural drainage have been more or less extensively worked already, so that, as a rule, any placer-deposits yet remaining undeveloped may be considered to offer some difficulty beyond the ability of the "old-timer" to overcome.

Fig. 1.


H
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HYDRAULIC LIFT
DESIGNED BY
W. R. Crandall, July, 1895.
(Patent applied for)
0 1 2 3 4 5 6
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AMERICAN BANK NOTE CO., N.Y.

This obstacle generally consists in lack of drainage, the deposit being often entirely below the water-level of the adjacent stream; and the remedy must be some means of raising the tailings and water to such an elevation as will allow their discharge into the natural water-courses.

This has been accomplished in California and elsewhere by means of the hydraulic lift or elevator; but the comparatively small areas available here preclude the use of the elaborate and expensive type of elevator hitherto used. Having been connected with an under-drain placer for the past two years, I submit (in accordance with a promise made during the discussion of the paper of Messrs. Nitze and Wilkens at the Atlanta meeting) a type of portable hydraulic lift or gravel-elevator, which I believe more nearly suits the requirements of the southern placer-deposits than any I have yet seen. The accompanying figures will perhaps explain the construction and operation of this device, as well as a more elaborate description. Fig. 1 shows the elevator in detail; Fig. 2, the manner in which it is set; Fig. 3, the details of the flume, etc. In all the figures the parts are lettered respectively as follows:

A. Cast-iron elbow at the base of the elevator.

B. Wings or vanes, to straighten the water before it enters the nozzle.

C. Nozzle.

D. Air-cap.

E. Air-inlet pipe, to furnish air when the bottom of the discharge-pipe is submerged.

F. Studs to support the discharge-pipe, and to keep it and the nozzle in line.

G. Cast-iron flanged throat.

H. Discharge-pipe.

I. Discharge-box.

K. Hood for discharge-box.

L. Adjustable wood-packing around discharge-pipe.

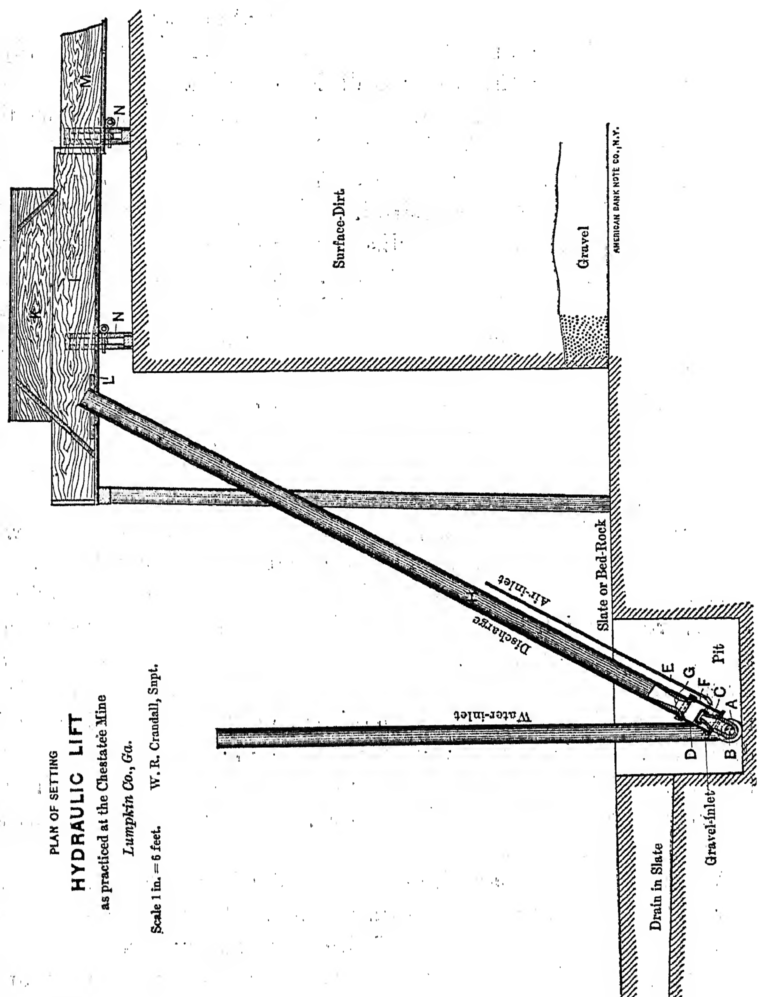
M. Discharge-flume.

N. Adjustable flume-supports.

This elevator, as used at the Chestatee mine, near Dahlonega, Ga., where it has been gradually developed and perfected under the needs of practice, consists essentially of an elbow, A, longer at one side than the other, and coupling by means of a flange

to a 5-inch pipe. At the other extremity are a flange, into which the nozzle screws, and three studs, F, which support the throat into which the gravel and water enter to be elevated.

Fig. 2.



PLAN OF SETTING
HYDRAULIC LIFT
as practiced at the Chestatee Mine
Lumpkin Co., Ga.
Scale 1 in. = 6 feet. W. R. Crandall, Supt.

The throat slips inside the 6-inch lap-welded pipe, H, for discharging into the flume on the bank, from which it may be conveyed wherever desired.

The whole apparatus, except the discharge-pipe, may be readily carried by two men. If it be necessary to move the elevator often, to keep up with the drainage, the portable character of the outfit is a great advantage.

At the Chestatee mine the practice is about as follows: The water-supply is conducted to the mine through a 9-inch pipe. At a suitable point the water is divided, and a 5-inch pipe conveys that used by the lift, while a 7-inch pipe conducts to the giant. Valves are provided at the Tee, so that one or both may be shut off as necessity requires. The lift is set into the slate to such depth as may be desired, and connection is made with the water-supply pipe. The discharge-pipe is then slipped over the throat and the discharge-flume is put in place, the discharge-pipe being set at such an angle of inclination as may be necessary to give proper grade to the tailings-flume and allow the pipe to extend a few inches through the bottom of the hooded box.

The air-pipe, E, is then screwed into place, and the lift is ready for operation. We govern the depth to which we set the lift into the bed-rock slate by the hardness or softness of the latter. If it be hard, frequent moving is cheaper than cutting slate-drains. If soft, we go as deep as the slate will stand without timbering. This we find to be about 7 feet.

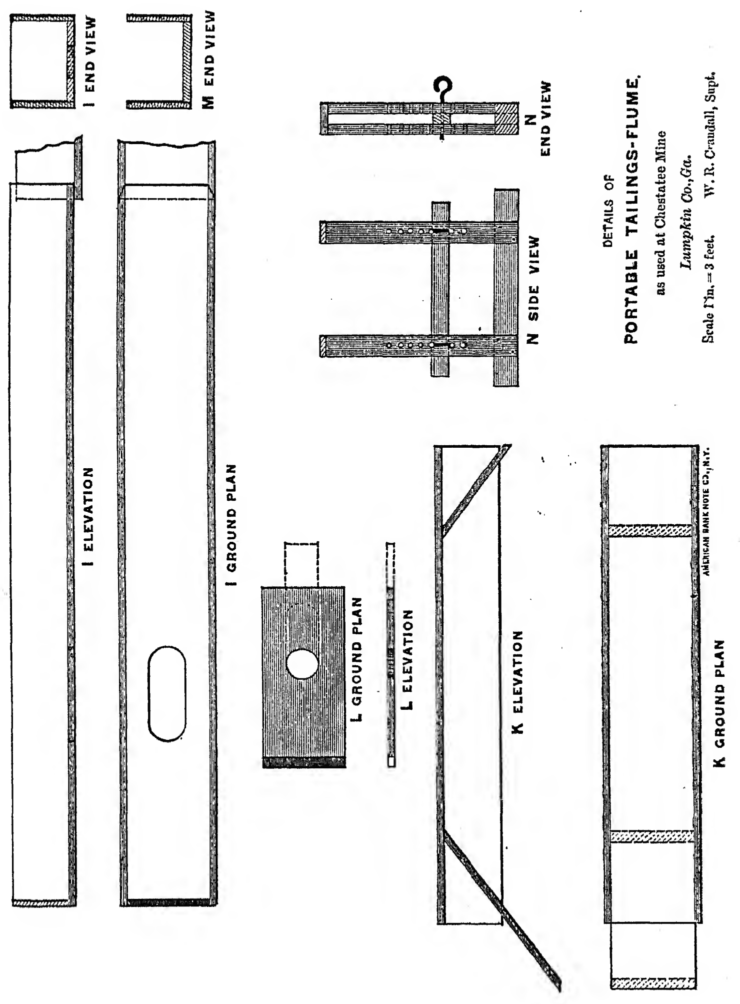
A main drain is then started in the general direction of our work, from which laterals are afterwards cut as required; and, at some suitable place near the lift-pit, a box about 6 feet long by 32 inches wide is set into the drain at grade, and in this is placed a cast-iron "grizzly" having round holes 2½ inches in diameter. This catches any rocks which may escape the forkers, and insures that nothing will get to the lift which will not readily pass through the throat, which has, when new, an opening of 3 inches.

We use straight-bar riffles in the discharge-flume, to catch any gold that may pass through the lift. This we find in practice to be about 5 per cent. of the total amount recovered—a result largely due to the fact that, when work is started at a new pit, the ground-sluices are not long enough to settle the gold thoroughly.

Whenever the drainage afforded by a pit is exhausted, the pipe-line is extended, a new pit is sunk near the gravel-breast and the work is continued as before.

As the work follows the general course of the river, the tailings are discharged into the river at the nearest point, the portable tailing-flume being extended far enough to insure the

Fig. 3.



safety of the immediate bank. The tailings finally flow through a ditch into the river.

We usually use about 200 feet of 5-inch pipe in the lift water-supply before extending the 9-inch pipe-line; and we

often move up 100 feet, dig the pit, re-set the lift and get ready for work again in one 12-hour shift with 5 men.

As to the work which the lift will accomplish, I may say that we are using a lift with $1\frac{1}{2}$-inch nozzle, discharging through a 3-inch throat into a 6-inch pipe, and lifting an average of 18 feet vertically, with water at about 60 pounds pressure per square inch.

As we are quite near the river, and have the drainage of a side-hill, the surface-water is considerable, probably 50 gallons per minute. We use a $1\frac{1}{4}$-inch nozzle on the giant, and the lift readily handles all this, and all the dirt and gravel we are able to wash to it. The latter we estimate, from measurements taken at different times, to be about $\frac{1}{2}$ cubic yard per minute of "topping." The quantity of gravel is hard to determine, owing to varying conditions; but it is safe to say that it is all that the amount of water employed will wash.

There is one other feature of the operations at the Chestatee mine which offers something of novelty, namely, that the water for both giant and lift is furnished by a water-actuated pressure-pump, used on direct-pressure.

With proper regulation as to the closing of the valves at terminal, so as not to run the pressure on the pipe-line beyond safe limits, this method offers some advantages: In case of need, a higher pressure may be obtained by the use of a smaller nozzle; the pipe-line and the whole water-supply are under closer control; ditch-watching and repairs are avoided; and when the pump is stopped there is no water available for the operations of casual outside prospectors.

In conclusion, I would say that I believe there are many places throughout our southern gold-fields where placer-deposits, hitherto considered inaccessible, may be worked at a profit by some modification of our practice here. Even where water-power is not available, steam raised with cheap fuel can be profitably used for the pumping, and the water can be employed under direct pressure for the giant or the lift, or both, and the simplicity and portability of such a plant as I have described render it especially suitable to this particular region.

The Ore-Deposits of the Australian Broken Hill Consols Mine, Broken Hill, New South Wales.

BY GEORGE SMITH, BROKEN HILL, N. S. W.

(Pittsburgh Meeting, February, 1896.)

THE Australian Broken Hill Consols mine is situated within a third of a mile eastwards of the famous Broken Hill Proprietory mine; but, so far as has yet been proved, the respective lodes have no connection whatever. The Proprietory line of lode, striking N.E. and S.W., is coincident with the bedding-planes of the inclosing gneiss, and, according to Mr. E. F. Pittman, A.R.S.M., government geologist, is undoubtedly a saddle-formation. This opinion, which met with much opposition at the time it was advanced, is now, I understand, being generally confirmed as the work of development opens out fresh areas for observation along the line of lode. The lode is of immense thickness—in places over 300 feet—and though it has been found to contain sufficient quantities of secondary silver compounds to yield phenomenal outputs of silver, its principal constituents are various lead and other ores, and it would appear more correct to regard it as an argentiferous lead-deposit than as a silver-lode proper.

The Consols lode differs from its gigantic neighbor in every respect. With an average thickness of not more than 18 inches, it has been worked along its course for upwards of 1300 feet, and at every point yet explored has been found remarkably well-defined and persistent. With a strike E. and W., it cuts obliquely across the bedding of gneiss and schist, continuing uninterruptedly through various bands of eruptive amphibolite. The dip, which is to the south, ranges from 24° near the surface to nearly vertical in the lower levels; but the alteration is not regular, as in places the lode is almost flat, and this at a vertical depth of over 300 feet. These changes of inclination have had no effect on the ore-deposits, which, as I shall endeavor to show, have been governed entirely by "cross-

veins" traversing the lode at different angles. It is only at the points of intersection of these "cross-veins" with the lode that important bodies of ore have yet been found.

The ore-deposits discovered up to the present time have been much scattered, and have consisted almost exclusively of silver-ores proper, the bulk of the metal being present in the form of stromeyerite and other permanent silver sulphides, which have been found to possess the same characteristics in whatever part of the mine they have been found. We may thus regard this as essentially a silver-lode, presenting the features of a fissure-vein.

All the ore yet won has been confined to those portions of the lode which are inclosed in the amphibolite, and the boundaries of this rock have been proved to be identical with the limits of the ore-bearing shoots. Where the metamorphic rocks have been intersected, the lode invariably pinches, sometimes showing no more than an inch-seam of fluccan. Three separate bands of amphibolite have so far been more or less explored, each apparently possessing the same peculiarities and possibilities, but of these only one has been extensively worked; and it is to the occurrence of the ore in this that I would direct attention, with no more than a passing reference to the others.

To a vertical depth of about 130 feet the lode-gangue is limonite, and, below this, siderite and calcite; these latter minerals often showing the banded structure characteristic of fissure-veins. In the zone of oxidation of the siderite, the removal of the calcite is practically complete; but where this oxidation is still incomplete and the siderite is altered externally only into limonite, the calcite is found in a partially dissolved condition, the solutions having penetrated some distance into the cleavage-planes, but not far enough to effect entire solution. The siderite having been the first of these two minerals to be deposited on the walls of the lode, it has (where it did not entirely fill the fissure) crystallized before the calcite was introduced. The removal of the latter mineral has, in consequence, left the upper part of the lode in a more or less vuggy condition, with very fine pseudomorphs of limonite on the hanging-wall; and on these, various minerals of subsequent deposition are occasionally found.

The gangue dissolving above has been carried downwards and deposited in the interstices of the lode, generally as a sludge, but sometimes showing incipient stages of crystallization. These changes can be seen still going on as the surface waters dissolve and carry down the more soluble minerals, the deposition readily taking place in open, undisturbed parts of the mine.

A similar process of alteration and removal has been observed with regard to some of the silver-minerals; but where the silver has been partially or entirely leached out, precipitation has apparently taken place in the immediate vicinity, in some instances probably on the very masses from which it had been so removed. This will be further explained with reference to one mineral which has been extensively altered, and which will come under notice in due course.

The ores of the mine exhibit many varieties, some of which have not been found elsewhere in the district, or, in fact, in Australia; but with four exceptions these rare minerals occur in small quantity, and it is unnecessary for the purposes of this paper to make extended reference to them. The following are the most important, and are named in the order of their productiveness:

	Containing silver. Per cent.
1. Stromeyerite,	about 30
2. Dyscrasite,	72 to 94
3. Antimonial silver chloride,	50 to 76
4. Fahlerz,	about 20

In dealing with the deposition of these minerals, I will confine my remarks to Nos. 2 and 3 as being the most uncommon and difficult of the series to account for under ordinary conditions of deposition; but before dismissing the others it may be interesting to note that Nos. 1 and 3 have not been met with in the lower workings, though each has been found at some distance below the water-level. Nos. 2 and 4 take the lead in depth, and each has been found scattered through the gangue in small quantities, ranging in size from grains to lumps weighing nearly 56 pounds. These small deposits have been found to assume a distinct track, and are evidently the continuation of the larger deposits worked in the upper levels, and deposited under conditions which I shall endeavor to explain.

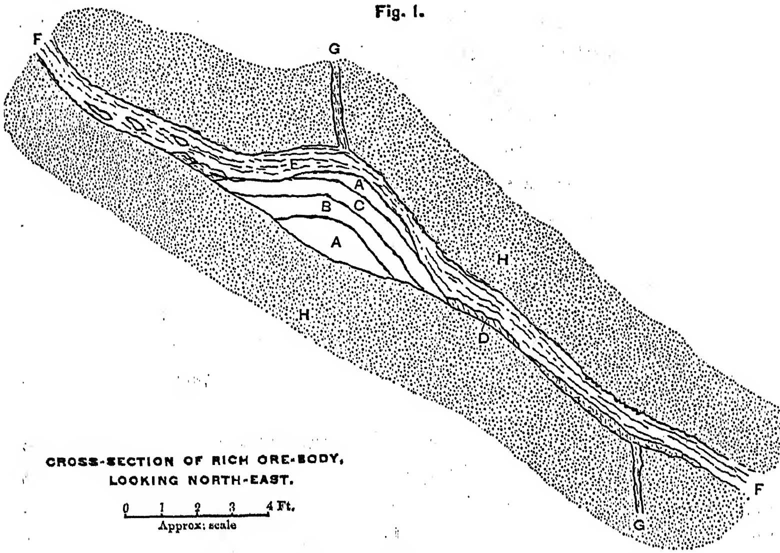
A peculiar fact in connection with the ore-bodies is their constant association with small quantities of cobalt-minerals. They are apparently inseparable; and, to the best of my recollection, neither has been found without the other. Where it occurs within the zone of oxidation, the cobalt-ore is generally more or less altered, and is then often argentiferous, sometimes to the extent of 5 per cent.; but, as a rule, it does not occur in immediate contact with the silver-ore, but in a separate vein of lode-material either above or below the latter. Below this zone the cobalt is almost solely in the form of cobaltite, and though it has been found in intimate mechanical mixture with the silver-ore, it is practically free from the latter when in its unaltered state.

The term "cross-vein," used herein, is adopted simply for convenience, and in preference to "indicator," a term which has been extensively used in Ballarat, Victoria, as referring to small bedded veins of slate or pyrites which have been found to strike across the auriferous quartz-lodes, leading to the discovery of important deposits of gold at the points of intersection with the lodes. The presence of these "indicators" may have had an effect upon the gold analogous to that which I believe the "cross-veins" of this mine have had upon the silver-deposits.

In that band of diorite (amphibolite) which contains the shoot most extensively worked, two separate and parallel "cross-veins" have been found; but, unlike the "indicators" of Victoria, neither is continuous. Their course is approximately N.E. and S.W., following the direction of the shoot, and wherever they have been met with, valuable deposits of silver have been found at the points of their contact with the lode. The larger of these falls vertically on the lode, and is composed of varying proportions of blende, pyrites, etc., with quartz; the thickness ranging from a mere streak to over 3 feet. By far the largest bonanzas yet found were in association with this "cross-vein." The accompanying sketch, Fig. 1, which was made by Mr. C. J. Klug, the mining manager, at the time the ore was being worked, shows that the deposit was made up of several layers of ore, etc., each remarkably distinct and totally different from the preceding one, indicating a great diversity and complete change in the successive solutions.

The smaller "cross-vein" is composed of biotite with clay, etc., but similar deposits have been connected with it, including several remarkable minerals, of which space forbids a description here. A very interesting discovery of a slab of dyscrasite must, however, be mentioned. This had been deposited in a vug in the siderite, the roof of which was perfectly crystallized; and the silver-ore, when deposited, had taken a complete cast of the crystals. On being freed from the gangue this specimen weighed 87 pounds, and presented a most interesting appearance.

Fig. I.



Australian Broken Hill Consols Mine, New South Wales.

A. Dyscrasite; B. Stromeyerite; C. Decomposed amphibolite, etc., assaying under 7 ounces per ton; D. Fahlerz; E. Soft gossany material, containing nodules of silver chloride, stromeyerite, etc., and averaging about 750 ounces per ton; F. Limonite, practically free from silver; G. Cross-vein; H. Amphibolite.

Both the "cross-veins" have been found to cut out and make again at irregular intervals for a considerable distance; but in depth both have been lost, and operations are now being directed in the lower levels with a view of picking up their continuation. The larger "cross-vein," which is shown in the sketch, has been faulted by the lode; and though it has been traced in an almost direct line for nearly 600 feet, it would probably be more correctly described as a succession of rock-joints formed along a line of weakness, and enlarged in places by a process of re-

moval and replacement. The biotite-vein is not so persistent; but it may have been of analogous origin, the minerals filling it being no doubt the result of a chemical rearrangement of the inclosing amphibolite.

In another part of the mine, 500 feet to the east, a separate shoot is being worked, which has yielded the same class of silver-compounds, deposited under similar conditions. This shoot is crossed almost at right angles by a veritable cross-vein of pyrites; and though this vein presents certain slight dissimilarities to those above referred to, its effect upon the silver-solutions appears to have been exactly the same; the ore occurring at the point of its junction with the lode.

It will thus be seen that wherever the cross-veins have been found to make junctions with the lode, valuable deposits of silver have been found, and no important find has yet been made except where a "cross-vein" has been in evidence. The lode-gangue is very often composed of most "kindly" material, which, as a rule, is practically free from silver (averaging less than half an ounce per ton), up to within a very short distance of the ore-bodies. It must, therefore, be admitted that whatever may have been the direct cause of the deposition of the silver, the cross-veins have played an essential part in the process.

The dyscrasite has been found in quantities ranging from the smallest of films and crystals, to huge blocks weighing over a ton; one piece, on being broken as small as possible for convenience in handling, weighed 16 cwts., and yielded fine silver equal to 80 per cent., the smelted value of which was over £4300 (1891). Another piece measured *in situ* 6 feet by 4 feet at its largest part, and averaged about four inches in thickness. The weight of this was about 23 cwts., but its silver value was rather lower. Altogether, over 6 tons of this mineral was taken from one deposit, yielding over 142,000 ounces of fine silver, together with other ores, principally stromeyerite, yielding an additional 335,000 ounces.

Practically the whole of the dyscrasite has been found crystallized; some of it, especially that occurring in calcite, being of great beauty. The most common varieties contained definite proportions of antimony and silver, as will be seen by the following analyses made of typical crystallized specimens:

	Silver. per cent.	Antimony. per cent.		
No. 1 =	72.9	27.1	agreeing with the formula . .	$Ag_3 Sb$.
No. 2 =	78.3	21.7	" " . .	$Ag_4 Sb$.
No. 3 =	84 4	15.6	" " . .	$Ag_6 Sb$.
No. 4 =	91.5	8.5	" " . .	$Ag_{12}Sb$.
No. 5 =	94.1	5.9	" " . .	$Ag_{18}Sb$.

The antimonial silver chloride is a specially interesting mineral, inasmuch as it carries with it certain evidences of alteration, from which much of its history can be gathered. All the deposits yet found have unmistakably shown it to be the result of alteration of dyscrasite. As I have previously observed, it is evident that the solutions which originally coursed through the lode must have been very varied. In a measure, the effects of each can be traced by its varying action upon those minerals that were most susceptible to its attacks. The particular solutions that have resulted in the mineral now under notice would seem to have contained certain quantities of lime, magnesia, iron, etc., as chlorides probably; and they would, therefore, be very similar to those at present existent in the mine.

I am indebted to the kindness of Mr. E. F. Pittman, government geologist, for the following analyses of this mineral, which were made by Messrs. Mingaye & White, analysts to the Mines Department. Unfortunately the samples analyzed are not taken from the purest specimens, but they were the best procurable at the time. The silver-contents are unusually low:

Analyses of Antimonial Silver Chloride.

	No. 1. per cent.	No. 2. per cent.
Moisture at 100° C.,	0.56	0.13
Combined water,	4.04	4.37
Silver,	47.46	45.87
Antimony,	16.87	20.72
Copper,	.11	.48
Lead,	.62	.31
Arsenic,	trace.	trace.
Gold,	trace.	......
Lime,	3.78	4.25
Magnesia,	1.17	.20
Ferric oxide,	2.11	.45
Chlorine,	13.69	12.27
Insoluble (gangue),	1.01	.90
Oxygen (by difference),	8.58	10.05
	100.00	100.00

It will be understood from these analyses that the alteration to which the dyscrasite has been subjected was very complete; and in effecting this change it would appear that the solutions, in causing the combination of a certain portion of the silver as chloride, had leached out a further portion which was immediately precipitated on the exterior of the mass as ordinary cerargyrite. The remaining antimonide has been oxidized by the same means into antimoniate, leaving the uncombined antimony behind as Sb_2O_4. Large lumps of this mineral have been met with, one weighing as much as 475 pounds. Many of the pieces, on being broken, showed the gradual action of the solutions, the outer crusts being completely changed, while the center consisted of a kernel of unaltered antimonide (dyscrasite) which the solutions had failed to reach. The intermediate parts often displayed the partially altered mineral, the change having advanced insufficiently to destroy the original crystallization. It is only above the unaltered siderite that this mineral has been found; and occasionally it is seen in close association with dyscrasite which, probably because the solutions were unable to reach it, shows no sign of alteration whatever.

Various deposits of small nodules have been found quite loose in open spaces in the lode, necessitating the not unpleasant task of shoveling them out. Each was spherical, and coated with cerargyrite crystals; these had evidently been contained originally in calcite as nodular masses of dyscrasite, which, upon the removal of the gangue, had been dropped out and acted upon as I have described.

The action of the solutions upon various other ores has been noted; and though the effects in some cases are most interesting, producing compounds of a very complex character, these cannot be touched upon here. The dyscrasite, however, appears to have been specially sought out for attack; and it has been noticed that where there was intimate association with fahlerz, the chloride was not formed, but the leaching-process was carried on in some instances to such an extreme that the whole of the silver was removed, leaving nothing behind but an insoluble white oxide of antimony. The quantities upon which such an extreme action has taken place appear to have been comparatively small, and generally the altered antimonide (dyscrasite) has been found as loose nuggets in the fahlerz, each

coated with the antimony oxide, and showing beyond all doubt that it had been considerably acted upon by solutions of some kind.

In attempting to account for the deposition of these and the other silver-compounds of this mine, I assume that the metals were originally brought from below in solution (this with all deference to Dr. Sandberger), in which state they would remain until the conditions were so changed as to admit of their deposition. It may be held that having been dissolved by weak solutions, under high temperature and pressure, they would be deposited on the removal of those conditions in the open portions of the lode. This is probably true of the ore-bodies of many mines; but in this case the *chemical union* of two elements of such weak affinity as silver and antimony, together with the constant association of the "cross-veins," would suggest the application of a special force passing through the latter. All the evidence yet produced in the mine's development points to an essential connection between the ore-deposits and "cross-veins," and it would therefore seem only necessary to ascertain what that depositing influence has been, to understand how the deposition was brought about.

Geologists are agreed that electro-magnetic currents are continually passing through the earth's crust, and, if so, these currents would, I presume, be far more likely to use separate veins as mediums for their transmission than to pass through solid rock, no matter what its character might be. If we bear in mind these two important facts, viz. (1) that the deposits are always connected with the "cross-veins," and (2) that electricity is constantly passing, it appears to me that we need go no further in our search for the depositing agent, which has been a continual succession of electro-magnetic currents passing through the "cross-veins," and causing the deposition—very often in remarkable combination—of the elements held in solution at the points of contact.

Unfortunately, the apparent absence of continuity deprives the "cross-veins" of much of their value as guides to ore-bodies; but, though they have not yet been proved to be continuous, I am nevertheless of opinion that they possess an unbroken connection of some kind which, in the present state of our knowledge, we are unable to trace. This connection may

be found in the ordinary rock-joints, but if so, they are too numerous and erratic to be followed with any degree of certainty.

The ore, as I have remarked, makes downwards in small quantities with more or less regularity, and, in the absence of any other known indication, is being followed in the belief that it will prove to be a connecting link between the large bodies, which will be found only where the conditions that were necessary to effect their deposition exist.

Such is the theory which the observations of several years have led me to form upon the occurrence of the ore-deposits of this mine; and my object in publishing it is to elicit discussion, and so lead others, who may have had similar experiences in other parts of the world, to relate them. The electrical hypothesis is advanced because I know of no other agency that is capable, under the conditions I have described, of producing these results. There are, no doubt, among my fellow-members many who could supply additional evidence on the subject, either confirmatory or otherwise; and I trust this paper may be the means of inducing them to do so.

Notes on Conveying-Belts and Their Use.

BY THOMAS ROBINS, JR., NEW YORK CITY.

(Pittsburgh Meeting, February, 1896.)

About six years ago the writer had occasion to visit a large magnetic iron-ore concentrating-plant, and then saw for the first time rubber belts employed for conveying-purposes. These belts were from 20 inches to 30 inches in width, and some of them were as long as 500 feet between centers. When I spoke of the enormous amount of material they handled with a small expenditure of power, the superintendent assented, but at the same time complained that although he bought the best quality of belts, the abrasion of the ore wore them out very rapidly, causing continually very large bills for repairs and renewals.

On close examination several interesting points were discovered:

1. It was noticed that the thin layer of rubber which covered the belt resisted the abrasion much longer than did a corresponding thickness of the cotton duck which formed the body of the belt; in fact, the life of the cover represented about one-half the life of the belt, although forming less than one-fifth of the total thickness.

2. Each layer or ply of duck wore out more quickly than the one preceding it, showing that the fibers were cut more easily when under tension, and, of course, the tension on each fiber increased as the number of fibers bearing the tensile strain diminished.

3. The wear was greatest in a line along the center of the belt. Frequently this part would be so quickly destroyed as to cause the belt to split in two longitudinally, though at the same time the portion nearer the edges was almost as good as new.

Noticing these facts, it became obvious that the functions of the cotton duck should be solely to give the belt tensile strength, and that it ought to be so protected by some abrasion-resisting cover that it would not be injured by contact with the material conveyed. It is also evident that this protecting cover ought to be of extra thickness over the center of the belt, in order to stand the harder work forced upon that part. Being engaged then, as now, in rubber manufacture, it was a simple matter to make a belt with a heavy rubber cover on the carrying side and thicker in the center than at the edges.

This reinforced cover renders the resistance to wear equal in all parts of the belt, and although being merely the anticipation of a patch, like the brass toe-cap on the schoolboy's shoe or the two-ply seat in his trousers, it was, like them, deemed patentable.

The ideal conveying-belt would be like the celebrated "one hoss shay," which disintegrated so evenly and completely when its work was done that there was nothing left to repair or regret.

Wishing to ascertain what particular compound of rubber would make the most durable carrying-surface, I made a lot of small samples, each mixed differently, and exposed them to a very powerful sand-blast, which in its effect approximated the conditions to which the compound would be subjected in actual

use, but it was more convenient for a large number of tests, being much quicker. The result of the first series of experiments indicated what grades of gum and what adulterants had better be left out, and also showed something that was very gratifying, namely, that there were certain adulterants which could be used in sufficient quantities to bring the cost down to a reasonable figure. I then made a second set of samples, following in the mixture the formula used in the more successful ones of the first set, but each new one was an attempt to improve upon its prototype. Some of the samples, owing to more intelligent methods in mixing them, proved so durable that the sand-blast test became too tedious, and a more severe and expeditious one was needed. This was found in exposing a disc of the rubber 6 inches in diameter by ⅛ of an inch thick to a heavy falling stream of crushed ore. The ore averaged about ¾ of an inch in size, and was delivered in a compact and heavy stream from the end of a very fast-moving belt. The sample was so fastened to a board as to receive the whole stream of ore and immediately deflect it. In this way the rubber came in contact with 200 tons of ore per hour, of which each fragment was delivered with considerable force full upon it. At first it was easy to see the comparative loss of weight after the sample had been exposed to the ore for an hour or two. In the next series results were very apparent after a day's run, but later, as results were developed which I was willing to accept as final, it became necessary to weigh each disc before and after the exposure, and thus learn the percentage of loss. The figures relating to the last set of compounds are as follows:

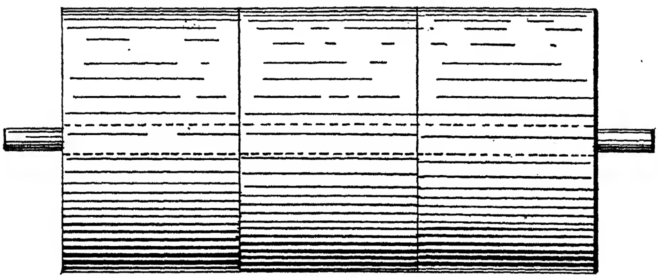
Samples.	Weight Before Test. In Grammes.	Weight After Test.	Percentage of Loss.
No. 154,	103	102.7	.0029
No. 155,	140	138.9	.0078
No. 156,	134.7	132	.0164
No. 157,	116.7	113.8	.0257

This test lasted for 12 hours' steady running under the conditions stated above.

Having at last decided upon the proper compound for the carrying-surface, I applied it to the belts, and I may say that every belt made since that time, which was in 1892, is in good

order to-day. In many cases, too, the belts which they replaced had been completely destroyed in three months' time under exactly similar conditions.

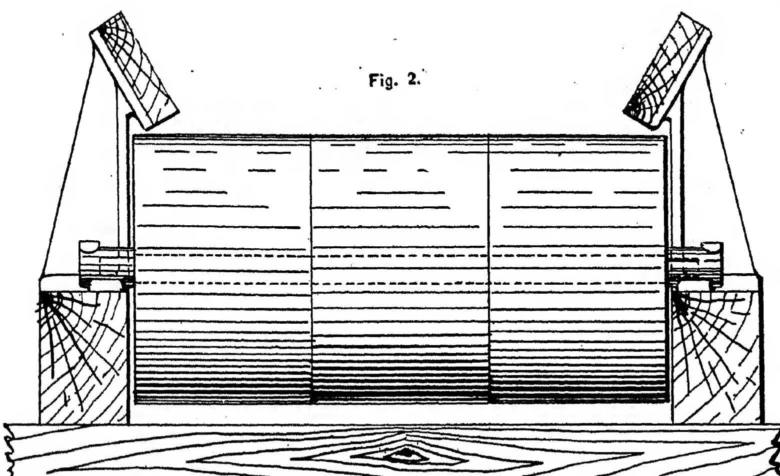
Fig. 1.



Oldest Method of Supporting Conveying-belts.

There are four principal methods of supporting conveying-belts.

Fig. 2.



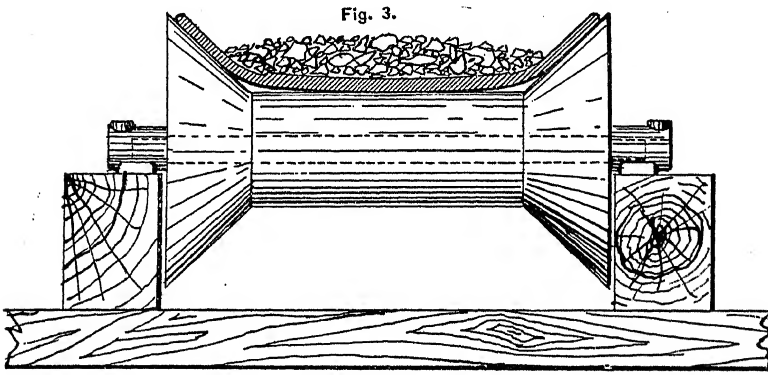
Idler-pulleys, with Skirt-boards added.

First we will consider the oldest method, in which the belt lies flat upon a straight-faced, horizontal pulley, as shown in cross-section, Fig. 1. On account of the liability of the material to roll off the belt, this form is only suitable under

certain conditions (*e.g.*, in carrying grain). The belt cannot be heavily loaded, and the feed must be so regulated that an even amount may be delivered to the belt at all times. If the material is below $\frac{1}{4}$-inch in size, the speed may be as high as 300 feet per minute. In carrying larger stuff on flat belts the speed must be lower; but the most necessary thing is to keep the belt very tight, that the material may not be jarred off in passing over the idler pulleys. This, of course, increases the strain on the bearings, and from that fact, together with their low efficiency compared with systems to be described later, we may consider flat-running belts as out of date for most purposes.

The second method (Fig. 2) is somewhat like the first, but

Fig. 3.

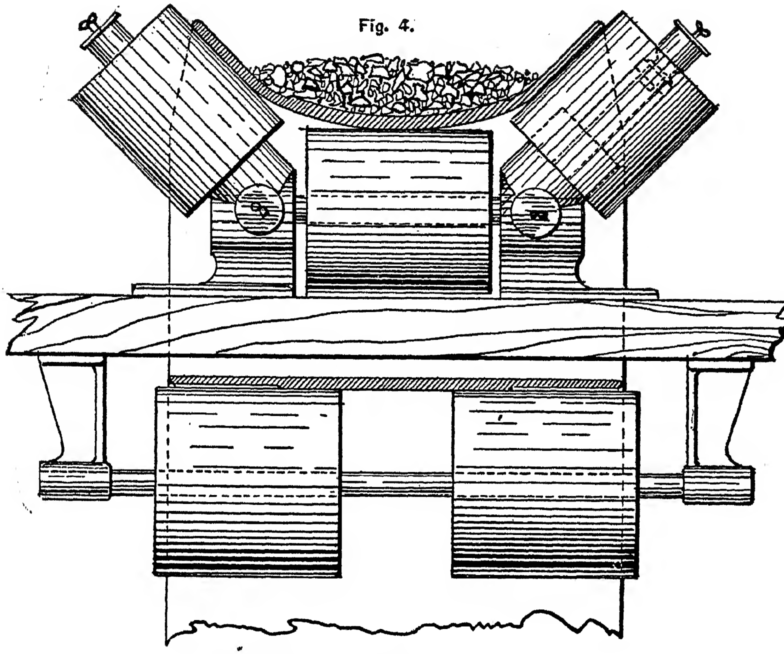


Conical Side-idlers.

with the addition of skirt-boards at the sides to increase the capacity of the conveyer.

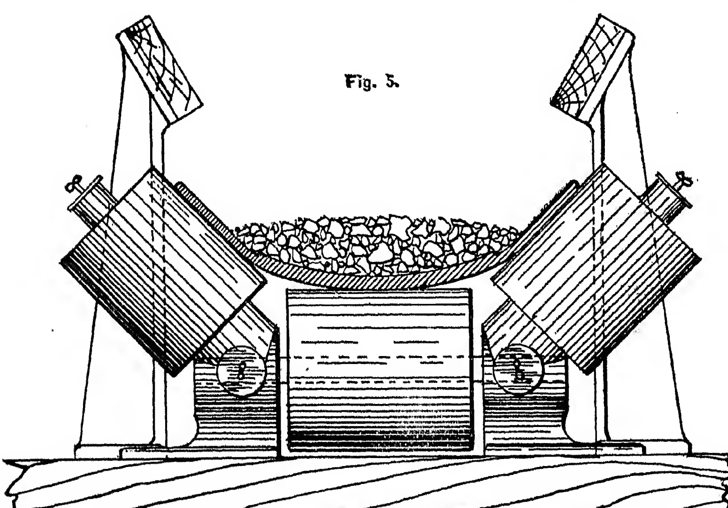
This method of rigging belt-conveyers is in great vogue among brick-makers and others who handle clay. It will be easily seen that the material must collect between the skirt-boards and the belt, and that, as it hardens, it will cut a strip off each side. The common practice is to start with a wide belt, and move the skirt-boards in as fast as these strips are cut off. When the width is so reduced as to render the conveyer totally useless, wheelbarrows are called into play until a new belt can be procured, and the entire process recommenced. This method is so entirely bad, that I refrain from further description. It is only fair to say, however, that the skirt-boards

Fig. 4.



Best Method of Supporting Conveying-belts.

Fig. 5.



Best Method of Supporting Belts with Skirt-boards.

fill one useful purpose, as it is the practice of the men shoveling into the belt to rap their shovels against the boards in order to get rid of the sticky clay. A board for this purpose can be applied, however, to a much better form of conveyer, and in such a way that it cannot interfere with the belt. (See Fig. 5.)

The third method is a slight improvement upon the last, in that a trough is made by raising the sides of the belt instead of using boards as described above. The conical pulleys used for this purpose are shown in Fig. 3.

This method has an obvious fault, owing to its bad design. By reason of the difference between the two diameters, the outer edge of the pulley goes twice as fast as the inner edge. This causes a slip which soon wears out the under surface of the belt. For belts not wider than 14 inches this form is not bad, for, with small belts, the weight on the pulleys is light, and the effect of the slipping is consequently less severe.

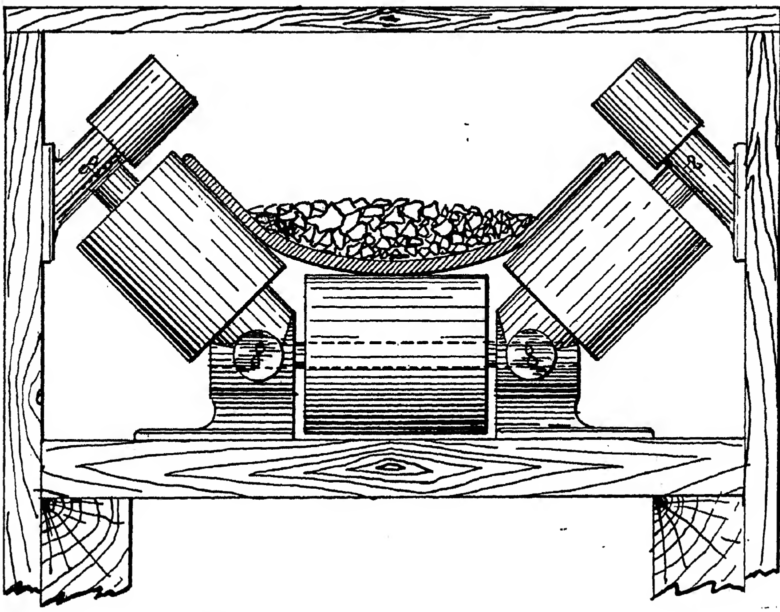
The fourth, and best form of belt-support, is composed of three pulleys, one carrying the middle or bottom of the belt, and one on each side with its axis at an angle of about 45°. The shafts of all three pulleys are held in a pair of combination bearings which can be adjusted to different widths of belt. I never supply any other form of support for belts wider than 14 inches, and when I refer to troughed belts in this paper, it is to be understood that the sides are raised by means of these angle-pulleys, one common form of which is shown in Fig. 4.

There has been no mention hitherto of the means of supporting the empty part of the belt on its return. This is done by a single flat pulley of the kind shown in Fig. 1, or with a pair of smaller pulleys with an interval between them, as shown in the lower part of Fig 4.

It is sometimes possible to save money in constructing a long conveyer by combining the first and last methods of belt-support referred to. If the belt were run flat the whole distance, it would need to be so wide that the extra cost of the belt would be about equal to the money saved in using the cheaper flat form of pulley; but by placing a set of troughing pulleys between every fourth and fifth set of flat pulleys, or at such other interval as may be found advisable, the load is so centred on the belt at each of these points that it has no time to overflow

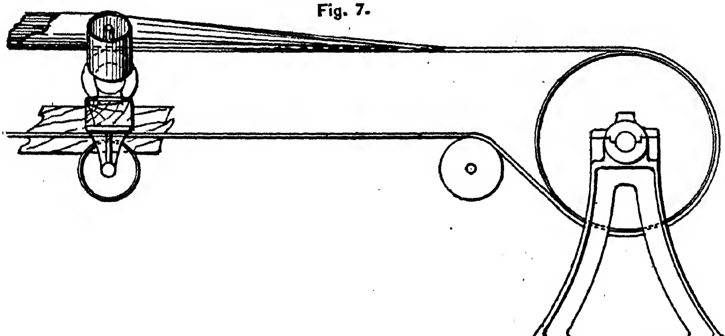
before it is again centered between the next pair. The use of a very wide belt thus becomes unnecessary.

Fig. 6.



Method of Fig. 4, with Guide-pulleys added.

Fig. 7.

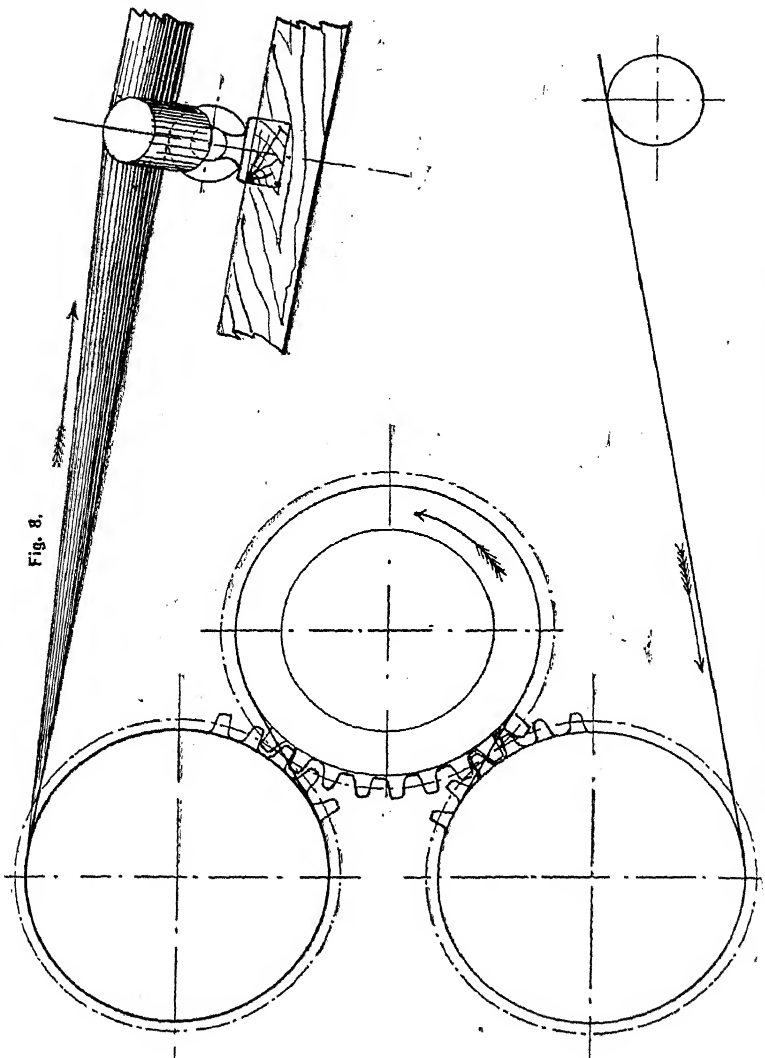


Side view, showing the Proper Position for the First Return-idler, in order to gain large arc of contact.

On some conveyers, it is often advisable to have at intervals a pair of idlers, running on a vertical axis, or inclined inward,

so as to make a right angle with the edge of the belt. These will serve to keep the belt straight on the pulleys if there is any tendency to run toward one side (See Fig. 6); but they are not needed with a good belt and strong, true supports.

Fig. 8.



Method of Driving Long Conveyers from the Receiving-end.

The large pulleys at the end ot the belt should be slightly crowned on the face, and the pulleys should not be less than 4 inches wider than the belt. The driving-pulley ought never to

be less than 30 inches in diameter, and in the case of long wide belts 48 inches is advisable, as it allows the first return pulleys to be so placed as to give the belt a very large arc of contact on the driving pulley. (See Fig. 7.)

Whenever it is possible, it is better to have the driving-pulley at the delivering end of the belt; but if it must be at the receiving end, a triple set of pulleys connected by gears can be easily arranged, which renders slipping impossible with the longest and heaviest loads. This scheme was first used by Mr. S. H. Edwards, Superintendent of the Magnetic Iron-Ore Co., at Benson Mines, New York. (See Fig. 8.)

The proper distance between the sets of idler pulleys is an

Fig. 9.

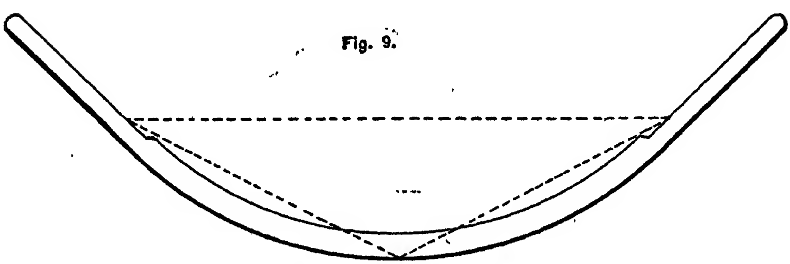


Diagram showing Method of Calculating Working-load.

important factor in the economical running of the belt, as will be referred to later. The troughing-pulleys should be from 4 to 6 feet apart, according to the weight of the load, and that for the return belt there should be pulleys placed under every alternate set of troughing-pulleys, which would make the lower pulleys from 8 to 12 feet apart.

To me, one of the most wonderful things in connection with this subject is the exceedingly small amount of power required to move enormous quantities of material. The power required, in one case, to run a conveyer, which carries 1000 tons per day a distance of 180 feet, and elevates it 40 feet while doing so, is all transmitted by a torn and frayed little 5-inch belt, which takes its power from a pulley on the shaft, and transmits it to a pulley alongside the head-pulley of the belt. The power is here divided, part of it going over a sprocket-chain to help drive a large dumping-apparatus. Yet the entire amount of power employed for both purposes does not exceed four horse-power.

It is impossible to give any rule for determining the exact number of horse-power required for conveying-belts running under different conditions, owing to the number of variable quantities which make up the load. One of the most important of the elements involved is based upon the distance between the sets of troughing-pulleys. If they are too far apart the belt sags down between, which materially increases the load. If, on the other hand, the idlers are too near together, the extra number of bearings makes another sort of resistance to be overcome. No general rule can therefore be made by which the required horse-power can be accurately determined.

It is a simple matter to determine the necessary belt-width and speed to perform certain duty when the weight per cubic foot of the material is known. If the belt is troughed it is safe to estimate that the load itself will cover one-half of the belt's total width, and that the depth in the center will be one-quarter of its own width. The area in inches of a cross section of the load (which we may consider an inverted triangle) will when multiplied by 12, give the number of cubic inches of material borne by the belt on each running foot of its surface. Multiply this result by some estimated speed to get the quantity in cubic inches that the belt will deliver per minute, and then reduce this to the number of feet, yards, pounds or tons delivered per hour, or to other convenient terms. (See Fig. 9.)

For example: To find the number of tons of material weighing 100 pounds per cubic foot that can be delivered by a 24-inch belt running 250 feet per minute:

As the belt is 24 inches wide we may safely consider the load as being a triangle standing on its apex and having a base 12 inches wide and a height of 3 inches. Therefore, the area of its cross-section will be 18 inches, and there will be $18 \times 12 = 216$ cubic inches on each foot of the belt. As the speed is 250 feet per minute, there will be delivered from the end of the belt 250 times 216 cubic inches, or 54,000 cubic inches per minute. This is equal to $31\frac{1}{4}$ cubic feet per minute, or 1875 cubic feet per hour, weighing 187,500 pounds, or about 93 tons per hour.

To save some of the steps referred to above, the following formula may be used. If we let $A =$ width of belt, then $\frac{3A^2}{8} =$

Table Showing the Number of Cubic Feet of Material Delivered in an Hour by Different Widths of Troughed Conveying-Belts Traveling at Various Speeds.

	Speed Per Minute in Feet is Shown in the Line Next Below This.													
Width of Belt in inches.	100.	150.	200.	250.	300.	350.	400.	450.	500.	550.	600.	650.	700.	750.
12	187.5	281.2	375.0	468.7	....	....	....	....	....	....	....	....	....	....
14	255.2	382.8	510.4	638.0	765.6	....	....	....	....	....	....	....	....	....
16	333.3	500.0	666.7	833.3	1000.0	1166.5	1333.3	....	....	....	....	....	....	....
18	412.8	619.2	825.6	1032.0	1238.4	1444.8	1651.2	1857.6	2064.0	....	....	....	....	....
20	520.8	781.2	1041.6	1302.0	1562.4	1822.8	2083.2	2343.6	2604.0	2864.4	....	....	....	....
22	630.2	945.3	1260.4	1575.5	1890.6	2205.7	2520.8	2835.9	3151.0	3466.1	3781.2	4096.3	4411.4	4726.5
24	750.0	1125.0	1500.0	1875.0	2250.0	2625.0	3000.0	3375.0	3750.0	4125.0	4500.0	4875.0	5250.0	5625.0
26	880.2	1320.3	1760.4	2200.5	2640.6	3080.7	3520.8	3960.9	4401.0	4841.1	5281.2	5721.3	6161.4	6601.5
28	1020.0	1530.0	2040.0	2550.0	3060.0	3570.0	4080.0	4590.0	5100.0	5610.0	6120.0	6630.0	7140.0	7650.0
30	1170.1	1755.0	2340.2	2925.2	3510.3	4095.3	4680.4	5265.4	5850.5	6435.5	7020.6	7613.9	8190.7	8775.7
32	1333.3	2000.0	2666.7	3333.3	4000.0	4666.5	5333.3	6000.0	6666.5	7333.1	8000.0	8666.7	9333.3	10000.0
34	1505.2	2257.8	3010.4	3763.0	4515.6	5268.2	6020.8	6773.4	7526.0	8278.6	9031.2	9783.8	10536.4	11289.0
36	1687.5	2531.2	3375.0	4218.7	5062.5	5906.2	6750.0	7593.7	8437.5	9281.2	10125.0	10968.7	11812.5	12656.2

number of cubic inches carried on each running foot of the belt. If a belt is run flat it will carry about one-third as much, or $\frac{A^2}{8}$. It is well to remember that when the width of a belt is doubled it will carry four times as much material, and when it is tripled it will carry nine times as much.

The table on page 89 will be found of service in determining the capacity of conveyer-belts.

In regard to this table I wish to say that the results shown are based upon a continuous and even delivery of fine material to the belt. As this is often unattainable in practice it is well to prepare for uneven and large pieces and for irregular feed by allowing a margin in either belt-width or speed, the two factors which govern the capacity of the belt. The widths of belting in most common use are 22, 24 and 26 inches, and the average speed is about 300 feet per minute. I am inclined to favor higher speeds, especially for elevating at an angle, as it seems to require less power to lift a small load at a high speed than the same amount of stuff per hour in a larger load at a low speed. A good speed for level work is 450 feet per minute; at an angle 650 feet is not at all too fast; and I have seen belts working smoothly at 900 feet per minute and at an angle of 27 degrees. Such speed as this, however, is hard on both the belt and idlers.

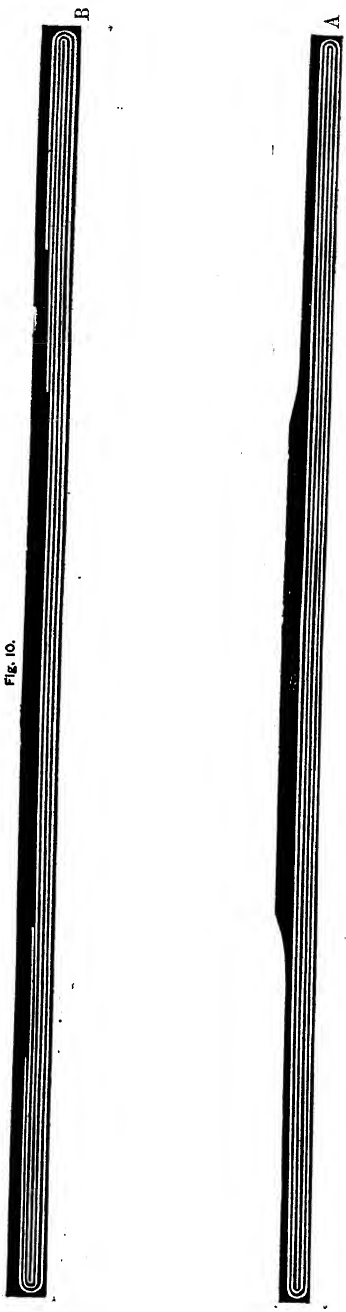
Having decided upon the proper width of belting for the duty to be performed the next points to be settled are the proper thicknesses for the belt and for its protecting cover.

The following table shows the suitable number of plies for different widths of conveying-belts:

Belts	20 inches wide and less	should	be	4-ply.
"	22 and 24 wide	"	"	5 "
"	26 and 28 "	"	"	6 "
"	30 to 36 "	"	"	8 "

The thickness of the rubber cover should be based upon the character of the stuff to be carried. For hard material weighing over 50 pounds per cubic foot the cover should not be less than $\frac{1}{4}$ inch in thickness. With the patent reinforced cover referred to above, it is possible to have this thickness at the center where it is needed, allowing it to taper off to $\frac{1}{16}$ inch or

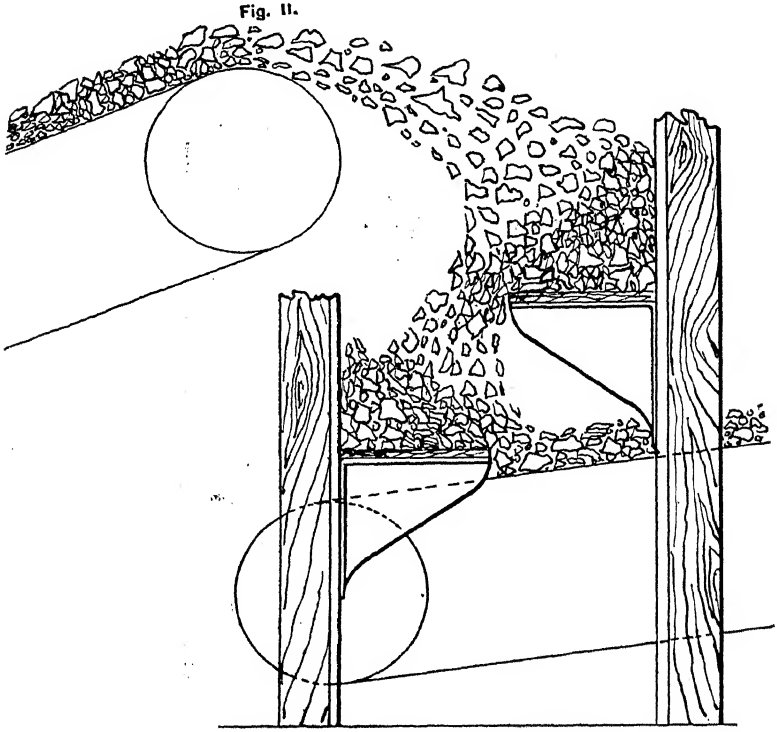
Fig. 10.



Cross sections showing Structure of Patent Conveying-belts.

$\frac{1}{8}$ inch at the sides where the work is lighter. This, of course, makes the cost lower than if the same thickness of cover extended the whole width of the belt. The belt with the reinforced cover referred to is shown in Fig. 10, A. There is now an improved form of this belt with stiffened edges, which make the belt bend properly at the center and preserve its troughed shape, while giving it such firmness at the edges that a mere

Fig. 11.



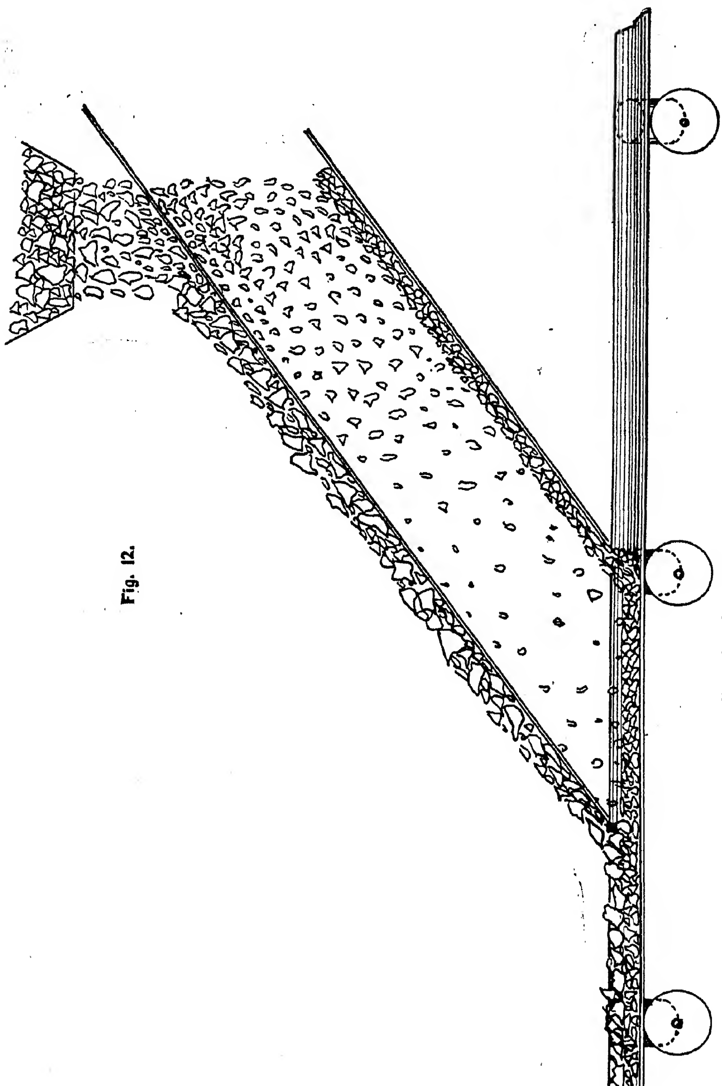
Section showing Delivery of Coarse Material from one Belt to another.

touch against the steering-idlers keeps it running true. This stiffening is done by running two or three plies of duck a part of the way in from the edges, giving all the advantages of a heavier belt but at a lower cost. This form will also be patented. (See Fig. 10, B.)

At the point where a belt receives its load it gets as much wear as in all the rest of its journey. A few points should be borne in mind in connection with this part of the con-

veyer. The material should not be allowed to drop vertically upon the belt but should instead be so guided by an in-

Fig. 12.



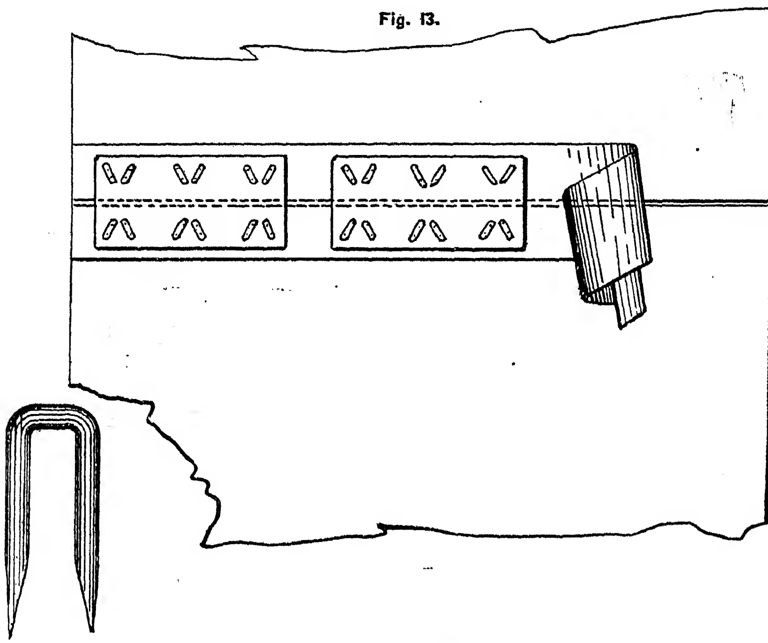
Combination of Grizzly and Belt.

terposed chute as to strike the belt with as nearly as possible the same speed and, of course, in the same direction as that of the belt itself. This is much the same principle as that which

causes a man who wishes to board a fast moving car to run along a few feet before jumping upon the step, except that in the former case it is the wear-and-tear upon the vehicle or belt which is avoided instead of injury to the passenger.

If ore or rock is carried, there is bound to be wear wherever it touches. It is well, therefore, if it has to fall any distance to let its force be broken by striking first against an accumulated

Fig. 13.



Method of Joining the Ends of Conveying-belts.

pile of itself and then roll off on to a chute, whence it may slide quietly upon the belt and not move until it reaches the end of its journey. See Fig. 11, which shows the device in use at the N. J. and Penn. Concentrating Works, Edison, New Jersey.

The chute should be made of cast-iron with sides, and should be so wide that the ore or other metal cannot get jammed, as it is very apt to do. The objection to a wooden chute is, that the broken stone or ore does not slide properly upon it, but runs down in a succession of jumps, that are as apt as not to carry it over the side of the chute or off the belt.

A device is shown in Fig. 12 for loading a belt under a crusher, from which falls both coarse and fine stuff. The line A represents a grizzly or screen of iron bars which allows nothing larger than 1 inch to go through. The small stuff slides down the chute, B, and forms a bed on the belt to protect it from the impact of the heavier pieces.

There are many ways of joining the ends of carrying-belts, the most common being by the use of metal belt hooks consisting of a set of teeth cast together with a metal back. These have to be destroyed if the belt needs taking up, and a better plan is shown herewith in Fig. 13. The thin steel plates are laid upon a strip of thin leather or canvas which covers the opening in the belt and thus prevent leakage. Staples are driven through the holes in the plate and clinched. In breaking the joint the staples are easily cut with a cold chisel but the plates themselves can be used an indefinite number of times.

The simplest method of lubricating the bearings is by means of compression grease cups which are dust proof and very economical. They are screwed into the ends of hollow shafts and a man without stopping can give the handles a turn as he walks along the belt, and this only has to be done once a day at most. Grease costing about four cents a pound is generally used.

Some of the purposes for which conveying-belts are used are the carrying of trap-rock and limestone in stone-crushing plants, charcoal and ashes in sugar-refineries, ore in concentrating-plants and mines, ore, fuel and flux in smelters, earth and stone in large excavations, blocks and logs of wood in pulp-mills, clay in brick-yards, moulding-sand in foundries, coal in breakers, yards, large power-plants and culm-piles, tobacco in process of manufacture, customers' packages in large retail stores, grain in elevators and flour-mills, boxed goods in coffee-mills, phosphate-ore in the southern mines and chemical fertilizer in plants all over the country. These are only a part of their uses, but the list of other purposes for which, though not employed, they are equally suited, and must some day be applied, would be a very much longer one.

Belts generally cost less to install than any link-belt or other metal conveyer; and the cost of maintenance is so much less that there is simply no comparison between them. In addition to this, they run noiselessly, instead of making a perpetual and

deafening racket, and no owner's ears can become oblivious to a banging and crashing of which every vibration means wear and depreciation, as well as loss of power.

You may say, "If this belt method of conveying is all that this man claims, why hav'nt we heard more about it? Why isn't it used more widely?" My answer is this: The various systems of sprocket-chain, metal-pan, trough- and screw-conveyers and the over-head cable-systems have all been owned by manufacturing concerns who make them known by advertising, undertake their construction and installation, and guarantee them. As to conveying-belts, no fundamental patent on the principle could be secured; and because a belt-conveyer is composed of two parts which are about equal in value and the products of entirely separate kinds of manufacture (I refer to, first, the iron pulleys and other supporting parts and second to the belt itself), no iron man felt like pushing the sale of a conveyer when he knew that some rubber manufacturer would share equally in the profits, and the rubber man would doubtless have felt the same if, with his less extensive knowledge of machinery, the thought ever occurred to him at all. And so, between them, the belt-conveyer has been neglected except where its merit compelled the recognition of a few wide-awake and self-reliant engineers. It has been the property of no one. Nobody's living depended upon its exploitation. It was to the interest of no one to stand sponsor for it; to develop, push it and perfect it. There are no data published about it, and each man who uses it to-day is practically the inventor of his own apparatus, which he has brought to its present state only after an expensive and vexatious experience. If an engineer to-day has a conveying-problem to solve and the thought of a belt occurs to him, immediately after come the questions: What kind of a belt? What width? How should it be run? On what? At what speed? Where can I find out? Being unable to answer any of these questions, and not caring to try experiments, his natural inclination is to state his needs to the well-known manufacturers of conveying apparatus and make a contract with one of them, thereby shifting the responsibility as soon as possible. The load of responsibility is apparently charged for along with the apparatus, but even the high price of the latter contains no suggestion of the great

and continual expense incurred in running and keeping it in order.

I do not wish to appear as condemning universally the use of all metal conveyers, for there are indeed some conditions for which they are better suited than are belt-conveyers. I may mention, for instance, places where the point of delivery must be constantly and quickly changed. With iron conveyers this is accomplished by the removal of a section of the iron trough, while a belt requires a movable frame-work containing two pulleys. This dumping-apparatus can be moved to any point in the belt's length, but it is neither as cheap in construction nor as quick in operation as the plan adopted with metal conveyers.

Nor do I condemn the great ingenuity which has made possible even the *poor performance*, by metal conveyers, of duties for which by their very nature they are unfitted. That is business. I only want to show that there are many places where other conveyers are now used for which belts could be substituted with economy.

The other day a man pointed to what he called a "real conveyer," which was dragging and scraping its load along (and apparently protesting against its own design and unfitness for the task), and observed: "Oh, this does the work cheap enough." It may be cheap, compared with manual labor; but it seems to me that with the competition of the present day nothing is cheap enough if there is anything cheaper. The object of these notes and sketches is to facilitate the use of belt-conveyers. I have gone into details, in order that any one who wishes to do so may construct a conveyer suitable for his needs. If there are any points which need to be simplified I shall be very glad to explain them to any member of the Institute who will write to my New York address, 71 Lexington Avenue.

Copper-Ores in the Permian of Texas.

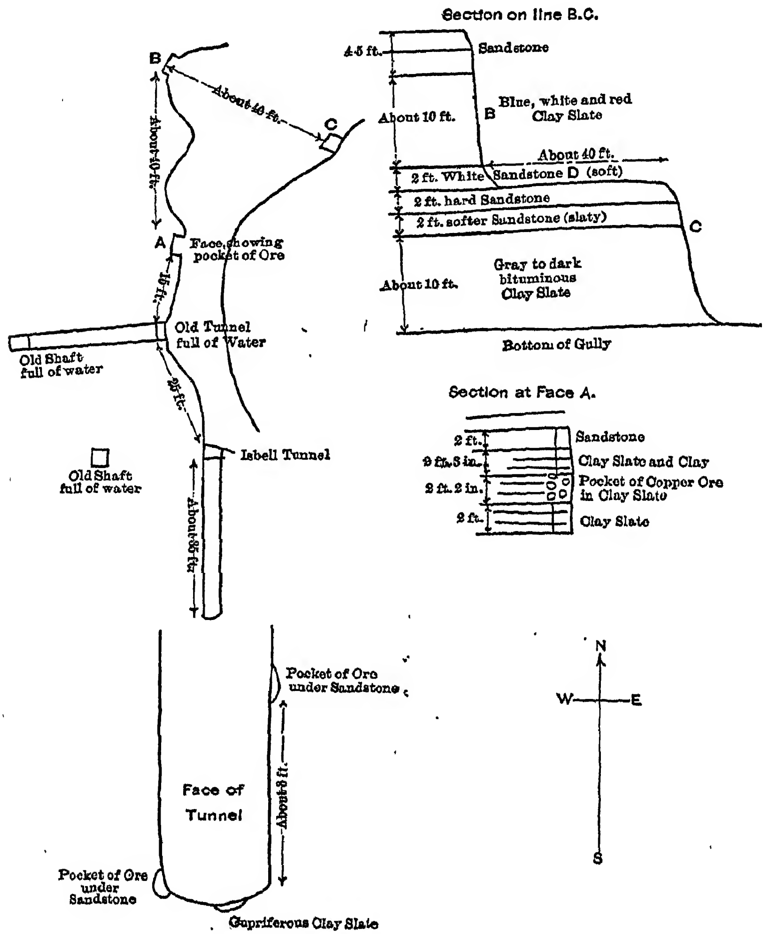
BY E. J. SCHMITZ, NEW YORK CITY.

(Pittsburgh Meeting, February, 1896.)

THE existence of copper-ores in the Permian measures of Texas has long been known, and these ores have been, from time to time, the object of geological researches and mining

developments. The most important of these efforts was made about ten years ago by the Grant Belt Copper Company of Texas, but it ended, after several years of fruitless labor, with an entire failure.

Fig. I



ISBELL LEAD

The ore appears principally in two zones of the Permian rocks, namely, the Red River zone in the counties of Archer, Wichita, Montague, Hardeman and Wilbarger, and the Brazos River zone in the counties of Haskell, Baylor, Stonewall and

Knox. The above-mentioned company prospected mainly through Hardeman, Haskell, Knox and Wilbarger counties.

From information collected by me, it seems that the geological adviser of the company assumed the copper-ore (or at least the copper) to be of plutonic origin, and was directing his efforts towards the depths for the mother-lodes or deposits. The diamond drill was employed for this purpose, and at one point, in Knox county, a hole was sunk to the depth of 1000 feet. How an engineer could conceive the idea that these copper-ores of the bedded Permian, which is bare of all plutonic lodes, dikes or intersections, must be of eruptive origin, is rather hard to understand. I have been told that indurated water-worn clay, mistaken for volcanic scoria, suggested or supported the hypothesis.

The Permian copper-ores appear in several horizons, and there exist two such horizons in each of the above-mentioned zones. In the Red River district, the lower horizon is reported near Belcher, in Montague county. It belongs to the lowest Permian, and lies not much above the line of contact with the underlying Coal-Measures. The upper horizon of the Red River district is represented in Archer and Wichita counties, etc.

The lower copper-horizon of the Brazos River zone appears in the counties of Haskell and Baylor, and the upper horizon in Stonewall county, etc.

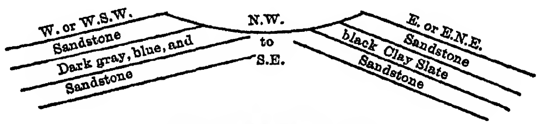
It should be remembered here that these horizons are not sharply bound to one continuous stratum or to the same level, as will easily be understood from the fact that the stratification of the Permian measures is somewhat irregular and non-persistent, and that the beds change rather abruptly.

Of the above-mentioned zones, that of Archer county, etc., is one of the most developed, and has been considered as one of the most promising territories; and having had occasion to investigate this district on a short trip during last summer, I hereby lay the principal results before the Institute.

The face of the country is very level, showing only small differences between the flat land and low hills, and the low bluffs (never more than 100 feet high) of the ravines and beds of Wichita river and its branches; the latter being principally the results of erosion during the wet season, while otherwise mostly dry.

The Permian measures consist of comparatively soft sandstones, clay-slates, clays, comparatively soft conglomerates and marls. The occurrences of copper-ore are scattered over a large area of Archer and Wichita counties, and the ore of Archer county appears principally in the marls and clay-slates as pseudomorph after wood (cuprified branches of trees, to a thickness of several inches in diameter), and as larger or smaller

Fig. 2

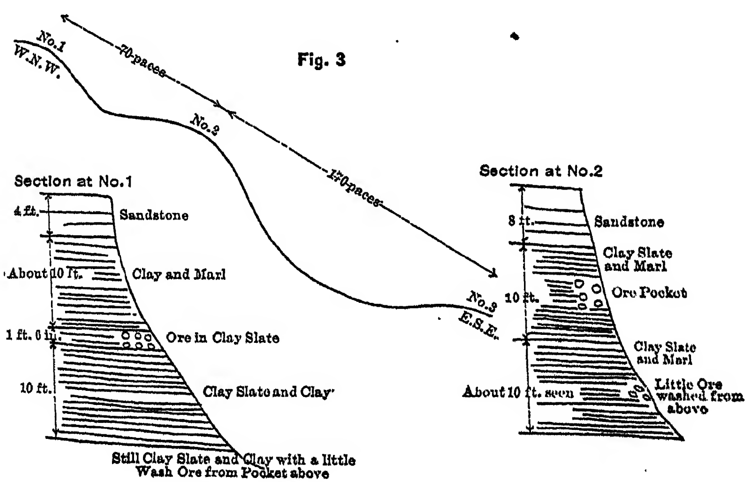


SECTION AT BALL MINE

nodules (up to 4 or 6 inches in diameter), most or all of which are of fossiliferous origin.

Copper-ore is found also in irregular amorphous masses, intermixing with and impregnating the marl or clay-slate. In a

Fig. 3



PLAN AND SECTION 1 MILE N.E. OF ISBELL TUNNEL

third form it occurs "as numerous small pebbles in a hardened cupriferous marl-conglomerate." And, finally, I found such nodules of copper-ore seated in hardened clay-slate and even in sandstone.

The copper-ore consists principally of green, blue and dark silicates and carbonates of highly varying percentage. The cuprified wood runs mostly high in copper, generally between 20 and 60 per cent., and the same is true of the nodules. When impregnating or intermixed with the clay and marl, the ore mostly contains less than 20 per cent. of copper; so does the conglomerate, etc.

No matter in what form the ore appears, it shows always its neptunic origin. The pseudomorphs of wood, as well as the nodule-ores, occur in entirely separate and distinct pieces of irregular form, and are scattered irregularly through the clay or marl matrix, forming nests or pockets of uncertain extent and size. The ore-occurrences in the conglomerate-marl and the cupriferous clays all show decided pocket-form, and give indisputable evidence of the origin of the copper-ores by precipitation during the deposition of the copper-bearing stratum, or by replacement and metamorphosis shortly after the deposition of the strata.

1. *The Isbell Lead.*—The so-called Isbell lead is located about 7 miles northwest of Archer City. The lead consists of a number of scattered copper-ore pockets and deposits which have been developed along the escarpment of a low hill and bluff, which follows a pretty uniform direction. The name "lead" may have been derived from the latter feature. See Fig. 1.

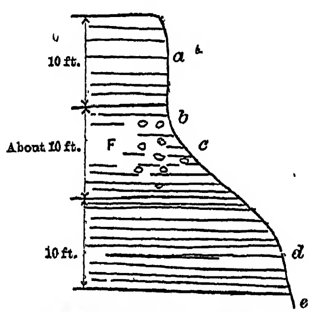
The ore occurs mostly as pseudomorph after wood or as irregular lumps of the dark and green silicates. It has its seat in a stratum of slightly bituminous clay-slate and marl. The ore appears also directly under and closely connected with the cap-rock, a sandstone, and small portions of the clay-slate and marl are also cupriferous. Of the old tunnels and shafts of the Isbell lead, ore was reported in the Isbell tunnel only; and I observed, after clearing out that tunnel, one small pocket of about 5 pounds of ore in the right corner of the face, and another smaller one of perhaps 2 pounds, about 8 feet back from the face of the tunnel and directly under or partly in the hanging-wall. In the lower part of the face I noticed a pocket of cupriferous clay-slate of about 10 to 12 inches in width, 6 to 7 inches in height, and 1 to 6 inches thick. Doubtless several other little pockets along the walls of the tunnel escaped my observation, as the walls had not been entirely refaced.

At face A, Fig. 1, I opened a little pocket of high-grade ore, which produced 32 pounds of ore from a space 2 feet 2 inches in height, 2 inches in width, and 1 foot 6 inches in thickness. An examination of the bluffs about 40 feet north and northeast of A found no ore, neither at face B nor at C. (See section on line BC, Fig. 1.)

The measures (clay-slate, etc.), cut at C, represent the same horizon and position as those in which the copper-ore pockets are found at A, and in the Isbell tunnel; and the ore would also be found at face C, if it were continuous and persistent in the bed of clay-slate and marl.

The general dip of the measures is slightly N.W.

Fig. 4.



SECTION ABOUT A HALF MILE S. E. OF ISBELL TUNNEL

a, reddish clay-slate; *b*, reddish clay, iron-ore, etc.; *c*, iron-ore in larger quantities, percentage of iron up to 40; partly conglomeritic; *d*, some blue slate, occasionally 1 foot; the rest reddish clay-slate, and conglomeritic iron-ore, etc., at the bottom, carbonate; *e*, reddish sandstone.

2. *The Ball Mine.*—The so-called Ball mine is from ½ to ¾ mile W.N.W. of the Isbell tunnel. The locality lies along the break of an anticlinal, and in the center are a number of old openings. See Fig. 2.

The ore appears also in a stiff, gray, white to dark bituminous clay-slate or marl as at the Isbell tunnel. The ore consists here principally of nodules and nuggets. A test-pit, 3 feet 8 inches in length, 2 feet 6 inches in width, and the same in depth, sunk in the center on the crest of the anticlinal into the clay-slate, produced 8½ pounds of good ore, and as about one-third of the nuggets and nodules may have escaped my attention, I may safely say that the above-mentioned space of 28 cubic feet produced 10 pounds of ore, which is not quite ½ pound to 1 cubic foot.

During a former period of development a trench had been cut across the anticlinal (W.S.W. to E.N.E.), but not quite to the bottom of the copper-bearing clay-slate, which is over 3 feet thick, and I could find nuggets of ore scattered here and there for the whole length of the trench, about 40 feet. I also noticed scattered signs of the ore for about 150 feet in a S.E. direction in the clay-slate until the same is covered by the sandstone; but my examination of the matrix to the N.W. (W.N.W. and W.S.W.) for over ¼ mile detected no ore.

Among the copper-bearing clay-slates brought from the test-pit I noticed a few fragments of rock, which apparently do not belong in this Permian clay-slate, and which may come from older formations. If such an origin was proved, it might be considered as a point in favor of the theory, held by some, that these copper nodules or nuggets are the result of a drift from older formations and a redeposition in the Permian strata. But these rock fragments occur so seldom, and the proofs against the drift-theory are so overpowering, that we may dismiss the matter without further consideration.

3. *The Winn Pocket or Deposit.*—This opening is from ¼ to ½ mile S.S.W. from the Ball mine. Here, also, the ore appears in a bed of marl or clay-slate capped by sandstone, which has been removed in a canyon along a local anticlinal. Next under the sandstone lies a marl-conglomerate, and below this the ore-bearing stratum which resembles more a marl than a clay-slate. Only the gray to dark varieties of the clay-slate or marl here, as in the other localities already mentioned, carry ore, while the red clay-slate and clay have no ore. At the Winn locality the reddish matrix forms about 90 per cent. of the marl-bed, so that only 10 per cent. of the beds are copper-bearing.

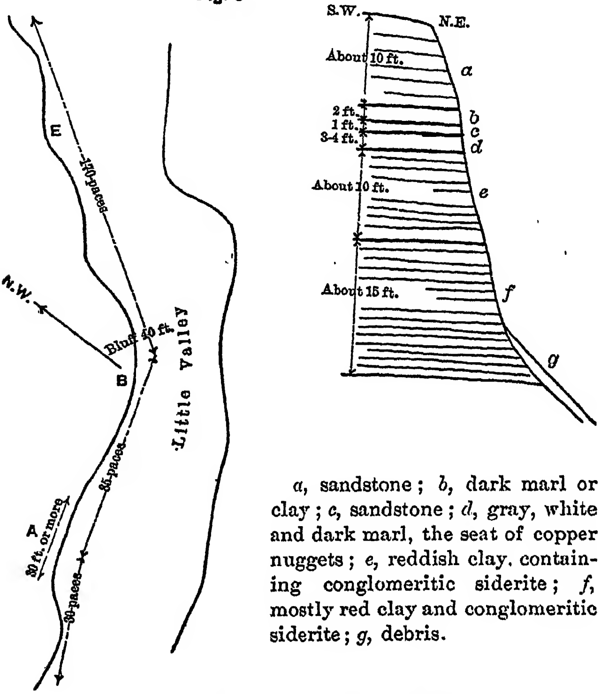
4. *The Elm Spring District.*—Around Elm Spring, which is about ½ mile west from the Winn locality and about 1½ miles S.W. or W.S.W. from Isbell tunnel, we have apparently the same horizon of the Permian as at the other localities already described. Elm Spring comes from a clayey marl schist capped by sandstone. The marl schist is a grayish-white material at the spring, and changes along the eroded bluff more to a clay-slate of gray, white, and reddish color. But no copper could be detected either in the clay slate or in the marl-schist.

Some copper-ore was collected from a mixed white and dark

clay-schist opened by some old diggings about ¼ mile N.E. from the Spring. It is principally nugget-ore.

And about 2 miles N.W. another outcrop of a small pocket appears in a bluish-white and dark clay under sandstone. A few copper-ore nuggets and pieces of sandstone crustified by green carbonate were observed.

Fig. 5



a, sandstone; *b*, dark marl or clay; *c*, sandstone; *d*, gray, white and dark marl, the seat of copper nuggets; *e*, reddish clay, containing conglomeritic siderite; *f*, mostly red clay and conglomeritic siderite; *g*, debris.

PLAN AND SECTION ON KERR'S FARM.

5. *Other Localities near Isbell.*—The sections in Fig. 3 show the Permian with the upper copper-ore horizon at a locality about 1 mile N.E. from the Isbell tunnel. This locality is between the North and Middle fork of Wichita river near the road to East's farm. Examining the clay-slate along the bluff I observed two ore-pockets between points No. 1 and No. 2, but no ore between No. 2 and No. 3. The pockets near No. 1 and No. 2 indicate from 200 to 500 pounds of good green ore.

From the above-mentioned locality, about ½ mile S.E., and

again for 500 feet at a point about $\frac{1}{2}$ mile south to S.W., I examined the bluffs 20 to 30 feet in height along the canyon, finding in both cases large clay-slate and marl below the cap-rock, but no sign of copper-ore. About $\frac{1}{2}$ mile S.W. the bluffs have the section shown in Fig. 4, and a height of about 30 feet. Here I noticed over a space of about 30 by 15 feet a large number of rounded, conglomerate, and flinty concretions, amounting to several tons, which were slightly cupriferous, but no good copper-ore was seen, nor was there any regular deposit of copper-ore. We are here about $\frac{1}{2}$ to $\frac{3}{4}$ mile S.E. or E.S.E. of the Isbell deposits.

The capping sandstone is missing at the section of Fig. 4, although we have here the same horizon as at Isbell, and the rounded concretions come about from level F.

In this locality Permian fossils (vertebrates and coprolites) are abundant.

About $2\frac{1}{4}$ miles west of Isbell workings and about 1 mile N.W. of Elm Spring I obtained the following section along the bluff:

1. Sandstone, lower part slaty, 2 feet.
2. Gray, bituminous clay-slate and marl, seat of the copper ore, 2 feet.
3. Purple sandstone with rounded edges, about 3 feet.
4. Variegated clays, containing siliceous concretions and siderite, about 25 feet.

In the débris, which covered the flat slopes or sides of the bluff and near the bottom, I found some samples of copper-ore which had been washed from above, and must have come from No. 2, although I could detect no ore outcropping in No. 2 along the face of the bluff. No. 2 is the same bituminous clay-slate as that observed at the Isbell and the Ball.

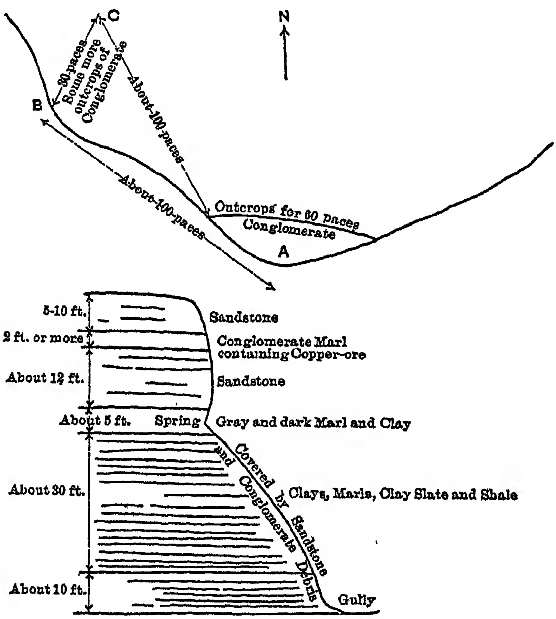
A short distance N.W. from the last section I found the capping sandstone removed from the clay-slate for an area of about 300 by 300 feet, by reason of a little anticlinal break in the formation. The break runs N.W. to S.E., and the underlying clay-slate was considerably weathered and washed and showed a large number of ore-nuggets scattered over this exposed area, and of which about $\frac{1}{30}$th to $\frac{1}{20}$th was covered with them. It is also possible that this ore continues under the cap-rock for some distance. So far as could be seen, the locus of the ore is

principally in the upper stratum of the bituminous clay-slate, but also to some extent in the lower (slaty) part of the cap-rock.

About 1½ miles S.E. of this locality, *i.e.*, about ½ mile S.E. of Elm Spring, I noted the following section:

1. Ledge of capping sandstone, about 10 feet.
2. Mixture of conglomerate iron-ore and rounded quartz-concretions in clay-slate or clay, 25 to 30 feet.
3. Mostly red clay, 25 to 30 feet.

Fig. 6



Plan and Section of Bluff at Spring Mountain, Texas.

Under this came the floor of a little valley, limiting the rock-exposure.

An examination made for about ¼ mile along the bluff detected no copper ore.

6. *Copper-Ore Pockets on Judge Kerr's Farm.*—The territory examined on this farm, 2½ miles E.S.E. from Archer City, on the road to Windhorst, is shown in Fig. 5. At A in the plan I observed an excellent showing of green copper-ore, pseudomorph after wood, located in the whitish-blue to dark-gray clays under the middle sandstone (at d) of the section. It ap-

pears along the face of the bluff in the same degree as the marl weathers and washes down. As already observed, I found a good showing of ore at A along the bluff; yet I have been informed from trustworthy sources that all the visible ore had been picked up along there only two years ago, so that the ore now apparent comes from that portion of the bluff-face which has weathered since. The copper-ore of the pocket at A covers about $\frac{1}{100}$th to $\frac{1}{200}$th of the surface. At E in the same plan the marl was found to be slightly cupriferous for several square yards along the face of bluff.

I observed also several samples of iron-ore (siderite) coming from the clay below the copper-bearing marl, which also showed the pseudomorphous form of wood, like the copper ore. This fact indicates an analogous formation-process for both ores under similar conditions, the iron-ore from iron-solutions and the copper-ore from copper-solutions.

7. *Spring Mountain.*—The third form in which copper-ore appears in the Texas Permian, namely, as cupriferous marl-conglomerate, can be observed in the Spring Mountain locality, which is about 2 miles S.W. from Arden. The cupriferous marl-conglomerate lies under a capping sandstone (see Fig. 6), and could be observed along the bluff of Spring Mountain for over 100 paces, from A to B C. The richest pocket of cupriferous marl appeared near A at the south end of Spring Mountain. It had an extent of about 60 paces, and the copper-ore percentage amounted to from 3½ to 5 per cent. or more of the total mass of cupriferous marl. The depth into the mountain could not be ascertained, but may have been 60 to 100 feet. The copper pebbles consist of siliceous green and dark carbonates, and are mostly very small, from ¼ to ½ inch in diameter. Dip of formation is slight, and runs from N.E. to S.E.

My examination along the road from Spring Mountain to Arden had very scanty results. Only a few samples of highly siliceous copper concretions could be discovered along the bluffs in an observation extending 1 mile or more.

Résumé.

The territory examined around the so-called Isbell lead extends about 1½ miles east and 3 miles west of Isbell and about 1½ miles north and 1½ miles south, therefore east and west 4½

miles and north and south about 3 miles. Copper-ore was observed in this district in at least 10 different localities, in the forms of pseudomorphs of wood, of nuggets, or nodules, and as cupriferous marl or clay-slate and clay. The ore appears always in irregular pockets of uncertain position in clay-slate and marl of highly variable thickness; but this clay-slate, etc., and the whole series of Permian measures inclosing the said clay slate and marl matrix belong undoubtedly in the same geological horizon.

In the localities 2 miles south of Archer City we have the copper-ore in the clay-slate and as cupriferous marl-conglomerate, both apparently of the same geological horizon with the deposits around Isbell.

The ore in the Texas Permian has this in common with the ore of the *Kupferschiefer* in the German Permian (Mansfeld district), that they both occur principally in a bituminous clay-slate and marl; but while the copper-ores of the former occur prevailingly as separate pieces, nuggets and nodules, collected or grouped to irregular pockets or deposits, the copper-ore of the *Kupferschiefer* is concentrated in one continuous thin layer of that formation. Moreover, the copper-ores of the Texas Permian are principally carbonates and silicates or siliceous carbonates; while those of the German Permian are mostly sulphurets, deposited in very fine particles.

But somewhat similar conditions must have existed during the process of deposition in both formations.

Coal-Dust as an Explosive Agent.

BY DONALD M. D. STUART, F.G.S., REDLAND, BRISTOL, ENGLAND.

(Pittsburgh Meeting, February, 1896.)

It gives me great pleasure to accept the invitation I had the honor of receiving through the Secretary of the Institute, to reply to the criticisms made in discussion of the theory advanced in my work entitled, *Coal-Dust an Explosive Agent;* and to lay before the Institute a further statement of the views which investigations have led me to form upon this subject,

published in my second book, entitled, *The Origin and Rationale of Colliery Explosions.*

I.—Reply to Criticisms.

From the report of the discussion upon the first book* I find that Dr. David T. Day, and Messrs. W. F. Durfee and William Kent felt some difficulty in accepting my statement, that there was not an appreciable quantity of carbon dioxide in the products of the explosions at the Camerton collieries.

Dr. Day said (p. 909):

"But, as a matter of fact, carbonic acid may have been present in considerable quantity, escaping notice because it was not carefully looked for. The only evidence of its absence is the fact that there was not enough to extinguish the lights of the exploring party, several hours after the explosion. The statement as to the absence of carbonic acid is therefore not well founded; and this is disappointing after the laborious citations of authorities on heat phenomena, gunpowder explosions, etc."

Mr. Durfee agreed with Dr. Day's general conclusions, and Mr. Kent remarked (p. 914):

"I think it a sufficient theory to suppose that the solid coal was distilled into hydrocarbon gas, ... that these hydrocarbons were burned in the ordinary way into carbonic oxide or carbonic acid, and that one or the other of these gases must have been there, although it was not found."

Again (p. 916):

"The attempt to discover CO_2 does not appear to have been made until eight hours after the explosion (p. 62), the full force of the intake current having been directed into the body of stagnant gases. The gaseous mixture that filled the roads was suffocating, pungent, and irritating. How would burning hydrogen produce such a mixture? It was probably CO, CO_2 and SO_2, with pyroligneous acid, or some such substance produced by partial distillation of the timbers. The failure to discover CO_2 by the appearance of the lights eight hours after the explosion, and after the full force of the intake current had been directed into the stagnant gases, does not prove that CO_2 did not exist immediately after the explosion."

These remarks were made under a misapprehension. It will be found at page 62 of the book, that four men in the New branch escaped, and in doing so, attempted to pass through over 150 yards of road in which explosions Nos. 8, 9 and 10 had occurred, and where the products of the explosions were

* *Trans.*, xxiv., 905.

imprisoned by the falls. They were in the stagnant atmosphere almost immediately after the disaster, and their candles burned brightly in it.

Upon the same page is recorded that:

"Stoppings were put in . . . . and the full force of the intake current directed into the body of stagnant gases in the Horse Road, with the result that Mr. Garthwaite reached the bodies about eight hours after the explosion."

In doing so he had to advance frequently into the stagnant atmosphere, and described it as "suffocating, pungent, and irritating." The record proceeds:

"Many times Mr. Garthwaite advanced more rapidly than the fresh air, getting into the stagnant gases, and invariably experienced the sensations named, necessitating immediate retreat to the air-current. On one occasion encouraged by his naked light burning clearly, he pressed forward in front of the current, and suffered loss of muscular power that caused him to stagger and reel, and he had to be assisted back into the fresh air. The candles of the exploring party are stated to have burned as brightly in the stagnant gases, as in the fresh air; and they could only discover in which atmosphere they stood, from the sensations they experienced in them."

The records therefore show that the exploring party were frequently standing in the actual gaseous products of the explosions, and this is not difficult to understand if the condition of the mine be recalled. The circulation of air was suspended, because the doors and crossings were destroyed, and the intake current escaped directly into the return airway, without entering the field of the explosions. The roads within the field were therefore filled with a still atmosphere, which comprised the products of the explosions and of the chemical actions in the antecedent spaces, with residual gases and normal exhalations of the strata. This still atmosphere could only be displaced by diffusion, because the roads were filled up by large falls at many points (described at pp. 21–24) which stopped the circulation. The exploring party entered into this atmosphere to make apertures through the falls, so that the intake-air with the motive column of the ventilation, could be made effective to force the stagnant gases along the roads to the upcast-shaft. They commenced this operation within two hours of the disaster, and were engaged for six hours, constantly pressing forward more rapidly than diffusion was effected, and were often standing in the stagnant gases; until, warned by physiological sensa-

tions, or by some member falling into unconsciousness, they were compelled to retreat to a point in the intake-air where the sensations disappeared.

The atmosphere resulting from the chemical changes in the field of disaster was, therefore, under constant test by the exploring party, for a period of six hours, at numerous points where the intake-air had not penetrated.

The usual method of testing the atmosphere of a mine for the presence of carbon dioxide is to observe the effect upon the flame of a lamp or candle; and where the flame burns clearly, it is known that carbon dioxide is not appreciably present.

It was known many years ago that an atmosphere containing about 2 per cent. of carbon dioxide extinguished flame. Dr. John Haldane and Mr. W. N. Atkinson (Inspector of Mines) recently investigated atmospheres in mines in which their lamps and candles were extinguished. They noted the physiological sensations experienced, and collected samples of the gaseous mixture for analysis. The results of their investigations are published in vol. viii., pp. 549–562, of the *Transactions of the Federated Institutes of Mining Engineers*, where it will be found that the atmospheres in which the lamps and candles were extinguished, contained from 2.21 to 2.58 per cent. of carbon dioxide; but that the first physiological sensation was experienced when the carbon dioxide rose to 3.26 to 3.38 per cent., and violent panting was felt when it formed 7.32 per cent., of the atmosphere.

The burning candles at the Camerton collieries show that the stagnant gases did not contain 2.21 per cent. of carbon dioxide; and the physiological sensations suffered by the exploring party were distinct in character from the sensations experienced by Dr. Haldane and Mr. Atkinson in atmospheres containing carbon dioxide. The evidence that there was not an appreciable quantity of carbon dioxide in the stagnant gases, is consequently indisputable.

Dr. Day further remarks (p. 910):

"A simpler explanation of the explosion would seem to me to be that the flame from the gunpowder set fire to the coal-dust in the presence of a considerable amount of air in the well-ventilated tunnel. This burning kept up along the tunnel, and was not very violent because the combustion was incomplete, and the

air and dust were not well mixed. Where the mixing was more complete, the explosions were more violent."

In such a mixture of coal-dust with "a considerable amount of air," the distilled gases must have been consumed as they were evolved, at the expense of the atmospheric oxygen surrounding each particle of coal; and no distilled gas could, in this excess of air, have been left to make an explosive mixture, as explained at p. 94 of the book. Dr. Day's hypothesis consequently fails to account for the principal phenomenon, namely, the exhibitions of explosive violence.

Dr. Day proceeds to say that "from such incomplete combustion carbon monoxide would be expected as the main product." Large volumes of carbon monoxide must, upon such a supposition, have been produced in the path of the coal-dust (about 1 mile long), and formed a considerable constituent of the imprisoned atmosphere. The survivors were in that atmosphere, immediately after the disaster, and the exploring party worked in it for six hours. They suffered the sensations recorded, but no fatal effects were produced. An atmosphere that contains only 0.5 per cent. of carbon monoxide is immediately fatal to human life; therefore, even that small quantity was not present, and Dr. Day's hypothesis fails in the positive absence of the proof he suggested as necessary for its acceptance.

Dr. Day continues (p. 910):

"Mr. Stuart ventures the statement that carbon dioxide is an essential product of the explosion of hydrocarbon gases, although it is well known that these products may vary from water and carbon dioxide as products of complete combustion, to mixtures in various proportions of hydrogen, solid hydrocarbons mixed with free carbon (soot), unburnt gaseous hydrocarbons, carbon in the form of coke, carbon monoxide, carbon dioxide, water, etc., according to the stage of incompleteness of the combustion."

The phrase, "explosion of gases," which I adopted to describe the cause producing the effects observed in wrecked and shattered structures, is perfectly understood as distinct from the inexplosive combustion seen in an ordinary illuminating-gas jet. Disruptive effects, like those recorded at the *loci* of the explosions, could only be produced by the explosion or practically complete combustion of the gas; and chemistry teaches that carbon dioxide is an essential product of that com-

bustion. If a gaseous hydrocarbon be mixed with its oxidizing equivalent of air and ignited, an explosion results, causing violent destruction; but if these proportions of gas and atmospheric oxygen be departed from, then, as their ratio for complete combustion disappears, the violent effects of the mixtures diminish until a point of incomplete combustion is reached in which no disruptive force is produced; and in this descending grade from complete to incomplete combustion, the products are necessarily changing, the carbon dioxide becoming a diminishing product. This suggestion of incomplete combustion, consequently, leaves the phenomenon of exhibitions of explosive violence unaccounted for.

Again Dr. Day observes (p. 910):

"We recognize it as impracticable to produce even experimentally any single product such as hydrogen (which Mr. Stuart claims) as the result of distillation, except by heating for a considerable time at the highest temperature attainable."

At pp. 46, 47 of my book hydrogen is shown to be one of the educts of the distillation of coal in association with other gases, and this result is obtained in ordinary chemical analysis. At p. 53 the educts are described as a mixture of gases, and the processes are discussed in which the hydrogen constituent of hydrocarbons became a free product in the gaseous mixture; that is to say, there was hydrogen present, no longer in chemical union with carbon, and therefore at disposal for explosive effects, where adequate oxygen could be obtained. This hypothesis is in accordance with the teachings of chemistry, and within the practical experience of the gas-engineer.

Dr. Day further remarks:

It is not necessary to use such a complicated distillation theory to explain the presence of soot, the common indicator of incomplete combustion, *i.e.*, a smoky flame."

The presence of the finely divided carbon would not be difficult of explanation, if it were an isolated phenomenon; but it was not. And every phenomenon must find its place in the *rationale* of the explosion. The chemical changes by which this carbon was produced, did not yield an appreciable quantity of carbon monoxide or carbon dioxide; consequently it could not have been the product of a smoky flame, which necessarily produces those gases. Combustible bodies in the path of the

explosions, which must have been consumed or burnt if this hydrocarbon flame had been present, were found neither consumed nor burnt, and exhibited a condition which was proof that there was not such a flame. It has already been shown that this suggestion of incomplete combustion leaves the disruptive phenomena unexplained.

Finally, Dr. Day observes (p. 910):

"As the explosion proceeded, its heat must, of course, have distilled some of the coal-dust; but it should be noted that after long heat-calculations Mr. Stuart admits that the slight amount of heat in the gunpowder would only account for a trifling percentage of the energy necessary to carry the gases to the place where he supposes the first explosion to have taken place. He admits that some combustion must have taken place, but he prefers to leave in the background this source of nearly the whole effect."

The last clause of this statement is entirely incorrect. It is shown at p. 85 of the book, that the surplus heat in the products of the mining powder was absolutely incapable of producing explosions at the points where disruptions were observed; but in the succeeding pages the source of supplementary heat is worked out, and at pp. 89 and 90 the quantity of heat is computed, and the figures are given, viz.: 44,104,476 gramme-units obtained in the oxidation of the hydrogen derived from the coal, and the hydrogen constituent of evolved hydrocarbons. The problem was grappled in the book, and an effort made to solve it.

The important criticisms of the theory advanced in my first book, have now been dealt with and shown to be largely due to a misapprehension of the facts, and therefore untenable; and the hypotheses advanced in substitution of that theory are found to be inconsistent with the phenomena of the disaster. It will now be an advantage to state briefly the observed phenomena and the explanation advanced in the book.

There were extensive residues of coked coal in the immediate vicinity of the shot (pp. 40 and 41), therefore the small quantity of heat at disposal from the charge of mining-powder, to which Mr. Kent draws attention (*Trans.*, xxiv., 915), was, as a fact, adequate to initiate distillatory action in the coal-dust.

The atmosphere of the roads for over one mile in length was laden with finely-divided carbon, and the enclosing walls were more or less covered with the same substance.

The products of the chemical changes in the field of the dis-

aster did not contain an appreciable quantity of carbon monoxide or carbon dioxide.

The first exhibitions of explosive violence were distant 136 and 140 yards respectively upon either side of the originating shot, and numerous disruptions were found at intervals beyond those points.

The explosions were local, the road being wrecked and the contained materials shattered to fragments in the immediate vicinity only.

In the spaces between the explosions, similar materials were neither damaged nor moved; and the road was uninjured except by the effects due to the immense changes of temperature to which the strata had been subjected during the chemical changes that occurred in them (pp. 77 and 78).

The explosions were in large sectional areas of road, or where the normal supply of air was greatly increased.

The explosive phenomena abruptly terminated in roads that were dry and dusty, but ventilated with feeble or "return" air-currents, and in wet roads.

These phenomena demand for their explanation the establishment of a series of chemical actions, in which heat was produced for its constant reproduction in the coal-dust in the the path of the current, up to the first explosions at the distant points named, without the appreciable production of carbon monoxide or carbon dioxide, and in which an explosive gas was accumulated for explosive combustion, which did not yield carbon monoxide or carbon dioxide as products.

The theory advanced in the book is, that the products of the mining powder were projected through the planes of ruptured strata to the floor, in fan-like sheets, setting up destructive distillation in the coal-dust; and the evolved gases flowed into the atmosphere of the mine, but could not undergo explosive combustion, although there was an ignition-temperature at command, because the atmospheric oxygen at that place was inadequate, or was not within reach, for complete combustion. With the limited supply of oxygen, a selective action took place, the free hydrogen yielded by the coal and the hydrogen constituent of a portion of the hydrocarbons seizing the oxygen for its oxidation, liberating carbon in solid form and in a fine state of division. The temperature reached by the oxidation

of hydrogen being about 2000° C., and there being a large quantity of heat generated in that action, the remaining hydrocarbons that had not previously undergone change, would at that temperature and in the absence of atmospheric oxygen, undergo dissociation, placing free hydrogen at disposal for disruptive action where oxygen could be obtained, and leaving more of the finely divided carbon in atmospheric suspension.

A series of chemical actions of constant sequence was, therefore, established in the immediate vicinity of the shot, in which there was the partial combustion of hydrogen referred to, generating heat, which produced a series of chemical changes identical with those which the heat supplied by the products of the mining powder had originated; and these regenerated activities were of constant and similar reproduction along the path of the coal-dust; the free hydrogen undergoing constant accumulation, until a place was reached in the workings of large sectional area, where the normal supply of air was greatly increased, and there being an ingnition-temperature in the partial combustion taking place, the accumulated hydrogen was oxidized with explosive violence, causing the disruptive effects observed at the *loci* of the initial gaseous explosions.

The quantities of heat generated in these explosions were much greater than the heat in the products of the mining powder; and it will be readily understood that an advancing series of changes similar to those described, was again established, producing explosions at points beyond, where the conditions for the oxidation of the accumulated hydrogen were fulfilled; consequently the explosions would be propagated along every path of the coal-dust so long as oxygen was available and there were no wet spaces to bring down the temperature.

This theory satisfies the demands of chemical changes causing a constant regeneration of heat, and placing hydrogen at disposal for disruptive action, without the production of carbon monoxide or carbon dioxide, and accounts for the absence of these gases from the atmosphere in the field of the disaster, with the fact that the carbon constituent of the hydrocarbons was not oxidized, but disseminated in that atmosphere, where it was found in copious suspension, and from which it had been deposited upon the vertical faces of the enclosing walls in dilated

layers, and in *striæ* giving the faces of stone and timber a veined appearance.

The chemical changes referred to as antecedent to the explosions, required time; and evidence may be asked as to the velocity of the gaseous movement between the *loci* of the explosions.

The food-bag found still hanging by its tape from the roof (p. 55), 13 yards from the shot, shows that there was very little velocity in the gaseous body at that point, though the evidences of chemical action abounded there.

The tram near the bottom of the Southeast Incline (Plate VII.), between explosions No. 5 and No. 6, was not displaced; but there were coked residues and copious deposits of fine carbon, the evidence of energetic chemical action (pp. 68, 69, 71, 72). About 60 pounds of lumps of coal, weighing from a quarter of a pound, to four or five pounds each, had disappeared from the open end of the tram (Figs. 34, 35, Plate VII.). If the coal had been removed by mechanical force, the lumps or their fragments must have been present somewhere; but they could not be found; and all that remained of them were the heavy deposits of coked coal distributed along the opposing faces of the two overhanging timber collars (Figs. 33, 35), showing that the lumps had been reduced to globules of coke. If there had been a high velocity in the horizontal movement of the gaseous body that attacked the coal, the particles of coke must have been swept away as they were produced; but they ascended almost vertically to the faces of the overhanging timber, indicating a slow onward movement in the activities. This phenomenon was between two propagated explosions, and over 300 yards from the originating shot.

These evidences show as a fact, that the gaseous movement along the path of the coal-dust was at a velocity that allowed time for the distillation of the coal and the remaining chemical changes in the series of actions.

The activities in the spaces antecedent to the explosions were therefore simple distillatory action, with chemical changes in the educts without the development of mechanical energy.

A phenomenon bearing upon this question was the local mechanical effect only upon the air-current (p. 96). During the occurrence of the explosions a man was standing at the

bottom of the downcast shaft, 946 yards from explosion No. 10 (the intervening road approximated to a straight tunnel), and he observed the momentary reversal of the flame of his open light once only, but it did not excite a suspicion that anything unusual had occurred. Another man was standing in the road at an intermediate point, 616 yards from explosion No. 10, and he observed nothing but one puff of wind that extinguished his light; but he attached no importance to it, as it was not an unusual occurrence, the intake air-current travelling at a high velocity.

The absence of mechanical effect upon the ventilating power of the mine again shows that the explosions were simply local, the disruptive forces dying out in all directions in their immediate vicinity; and that there was no appreciable mechanical force in the gaseous movement in the intervening spaces. This phenomenon is not difficult to understand when it is remembered that the gaseous hydrocarbons that were being constantly added to the atmosphere were almost immediately eliminated by the oxidation of the hydrogen constituent; that the volume of the incoming air was being constantly diminished by the removal of the oxygen, forming one-fifth of that volume, for the oxidation of the hydrogen in the explosions and antecedent actions, in successive productions of steam undergoing eventual liquefaction by surrender of heat; and that, by means of this exhaustion, the mechanical equilibrium of the air-current was maintained.

The abrupt terminations of the explosive phenomena in the paths of the coal-dust require explanation (p. 75). The fundamental difference between the path of propagation and the path in which propagation failed, was the condition of the air-currents. The former contained "intake"-air; the latter, with the exception of a trifling split, were supplied with "return"-air, impoverished of its oxygen and laden with oxidized products.

The explosive phenomena were also arrested in wet spaces, which presents no difficulty when it is remembered that the product of the oxidation of hydrogen would, by its contact with wet surfaces, undergo a change corresponding to what takes place in the condenser of a steam-engine. The external stratum of the product coming in contact with the cold wet

surfaces would be instantly condensed, leaving a vacuous space, into which another stratum would fall and be condensed; and the process would be repeated so rapidly during the slow gaseous movement, as to make condensation practically instantaneous. The product being liquefied by surrender of heat, the temperature would fall below the distilling-point, and the chemical actions would come to an end.

The arrest of propagation is consequently brought about by atmospheric conditions, in which heat cannot be generated for the reproduction of the chemical actions because there is inadequate oxygen to sustain the chemical changes in which the heat is produced; also by wet surfaces, which demand the surrender of the heat for the evaporation of their moisture.

All the observed phenomena from the origin to the termination of the disaster, therefore, find their interpretation in the theory advanced in the book, *Coal-Dust an Explosive Agent*, to which I must refer for the full records and their investigation. The theory, shortly stated, is: A colliery-explosion in which coal-dust is the principal agent, comprises numerous local explosions, separate in time and in space, at irregular intervals, where the normal supplies of atmospheric oxygen are greatly increased, and caused by the explosive combustion of accumulations of hydrogen gas, derived from the coal-dust in the antecedent spaces, by a series of chemical actions of constant sequence, which produce heat for regeneration without auxiliary intervention, and are constantly reproduced along the path of the coal-dust under the conditions named.

This theory accounts for the coked residues of coal and the positions in which they were found (pp. 40, 41, 42, 69, 94, 95); explains the presence of amorphous carbon in atmospheric suspension, and its characteristic deposition upon the inclosing walls throughout the field of disaster; accounts for the practical absence of carbon monoxide and carbon dioxide in the gaseous products; supplies the heat for the distillatory action, for raising the air of the gaseous mixture to ignition-temperature, and for supplementing the losses due to the contact of the products of combustion with the cold surfaces; allows time for the distillation of coal and the chemical changes in the series of actions; provides the explosive gas for producing the disruptions; explains the occurrence of the disruptions at inter-

vals, the absence of mechanical effects in the intervening spaces, the conditions under which propagation proceeds or is arrested, the maintenance of the mechanical equilibrium of the ventilating current, and excludes no coal except that which yields no combustible gases when subjected to the temperature of destructive distillation.

II.—The Origin and Rationale of Colliery Explosions.*

The views in the first part of this paper were founded upon the phenomena of the explosion at the Camerton collieries, which provided a practical demonstration of disaster caused by gas derived from coal-dust; but it was reasonable to suppose that evidence beyond a solitary explosion of that character, would be demanded for the establishment of a new hypothesis. Supplementary evidence is now provided by the explosion of February 9, 1895, at the Timsbury collieries, Somersetshire, which had been worked for the previous seventy years with candles (which is the mode of lighting still employed). There has never been a trace of fire-damp observable in the collieries. Facilities were afforded me for investigating this disaster, and I found its phenomena to be identical with those of the explosion at the Camerton collieries, and the two disasters to provide an important body of evidence for investigating the subject of colliery-explosions. The result of the investigations is given in my second book, in which the views just considered are found to be confirmed.

The first 69 pages are devoted to the elucidation of the explosion at the Timsbury collieries. The conditions are described upon pp. 4–10, where the absence of fire-damp is considered, the method of working and ventilation explained, and dry coal-dust shown to prevail in the workings generally. The phenomena of the explosions are recorded on pp. 11–36. At

* *The Origin and Rationale of Colliery Explosions*, founded upon an examination of the explosions at the Timsbury, Albion, Malago Vale and Llanerch Collieries, and upon the principal phenomena of the disasters at the Abercarne, Alltofts, Altham, Apedale, Blantyre, Bryn, Clifton Hall, Dinas, Elemore, Hyde, Llan, Mardy, Morfa, Mossfields, National, Penygraig, Risca, Seaham, Trimdon Grange, Tudhoe, Udstone, and West Stanley Collieries. By Donald M. D. Stuart, F.G.S., Mining and Civil Engineer; author of *Coal-Dust an Explosive Agent*. Bristol: John Wright & Co. London: Simpkin, Marshall, Hamilton, Kent & Co., Limited. New York: Hirschfeld Bros.. 65 Fifth Avenue.

Lower Conygre shaft, Plate I., the roads were found in their normal condition after the explosion (p. 12), but 230 yards distant some débris was lying upon the floor from a shot that had been very recently fired. The double timber in the vicinity of the shot was undisturbed (p. 13). A food-bag, 34 yards distant, was still hanging from the roof by its tape in a carbonized condition; empty trams in the siding near were uninjured and undisturbed; more trams at the foot of Peter's incline, beyond, were in the same condition; and the man who charged the hole with mining powder and lighted it, was found there, burnt, but not mutilated. At the top of this incline, however, the empty trams were destroyed; one wheel, of 12 inches diameter and 2 inches tread, was broken in two, and the parts were lying in opposite directions; the iron drum was broken, and its timber staging shattered; the ventilating doors were reduced to small bits, and the brick and stone arching was wrecked (pp. 14, 15). A violent local explosion had occurred here, and the field of disaster contained many similar local exhibitions of explosive violence at various intervals.

At Peter's incline the road divided (Plate I.): one branch to Wyatt's incline reached several districts of workings; the other branch proceeded to Upper Conygre shafts; and roads from other districts joined in at intermediate points. The initial explosion was propagated into each district, and no less than eighteen disruptions were observed at widely distant points, described in pp. 15–28. A summary of these disruptions is given at p. 30 as follows:

					Yards.
From the shot to No. 1 explosion with the air-current,					191
"	No. 1 explosion to No. 2 explosion against the air-current,				123
"	No. 2	"	No. 3	" " "	117
"	No. 2	"	No. 4	" diffusion alone exerting a current,	66
"	No. 4	"	No. 5	" with the air-current,	230
"	No. 5	"	No. 6	" " "	45
"	No. 4	"	No. 7	" diffusion alone exerting a current,	266
"	No. 1	"	No. 8	" with the air-current,	165
"	No. 8	"	No. 9	" " "	263
"	No. 9	"	No. 10	" against "	61
"	No. 10	"	No. 11	" " "	88
"	No. 11	"	No. 12	" " "	70
"	No. 9	"	No. 13	" with "	187
"	No. 13	"	No. 14	" " "	38

						Yards.
From No. 14	explosion to	No. 15	explosion	against the	air-current,	100
" No. 15	"	No. 16	"	"	"	62
" No. 15	"	No. 17	"	"	"	77
" No. 17	"	No. 18	"	"	"	170

All the disruptions were in large places or where the quantity of air was abnormally large.

Where the propagation of the explosions failed in the path of the dry coal-dust, the roads were supplied with impoverished and vitiated air-currents (pp. 65–67).

Coked residues of coal were found in the vicinity of the shot and at numerous places in the workings. In the spaces between the explosions, the residues were near the floor, and feathered off into small patches at heights not exceeding 2 feet above the rails, where they were deposited upon obstructions (*e.g.*, trams) standing in the path of the coal-dust. There were two exceptions: coal-dust, resting upon a plank 4 feet above the floor, was found coked *in situ*, and some timber at the end of a level was generally covered on one side (pp. 31–33). The mode of deposition is considered at pp. 51, 52. Where the coked residues were within the influence of the disruptive forces, they extended to the roof.

The stagnant atmosphere of the roads in the field of disaster was so charged with amorphous carbon, that it was compared by two members of the rescue-party, who first entered the roads, to the condition that is produced "by turning a bed-tick inside out and shaking it in the air." This carbon was universally deposited upon the walls of the roads (pp. 34, 35).

The candles burned clearly everywhere in the stagnant atmosphere, but the rescue-parties suffered smarting of the eyes and nose and loss of power, and were frequently compelled to retreat to the pure air (p. 35).

The origin of the disaster is investigated at pp. 37–40, and traced to the shot referred to; and the circumstances of the shot-firing, the quantity of mining-powder employed, the heat-energy at disposal after the strata were ruptured, the evidence of watering the road, and its condition at the time of the disaster, are considered at pp. 40–47; where it is shown that the charge of explosive was not less than 12½ ounces; that there was a surplus of about 137,356 gramme-units of heat for dis-

tillation, and that the coal-dust at the shot was in favorable condition for that action.

The links of evidence of distillation at the shot, the food-bag, and the bottom of Peter's incline, and the unbroken deposition of amorphous carbon between the shot and the explosion at the top of that incline, show the constancy of the chemical changes between the two points.

Considering the large quantity of heat at disposal from this initial explosion, its propagation along each path of the coal-dust so long as adequate atmospheric oxygen could be obtained, presents no difficulty, as it simply demands a reproduction of the chemical actions that are seen to have had a place between the shot and the initial explosion.

The conditions prevailing from the shot in the contrary direction towards Lower Conygre, were unfavorable to the reproduction of the chemical changes in that direction (pp. 48, 49). The floor was wet, and the first door formed a *cul-de-sac*, in which the still atmosphere was saturated with water-vapor. The heat was consequently exhausted for vaporization, and distillatory action failed.

The undisturbed timber, the undamaged and unmoved trams, the hanging food-bag, the absence of mutilation of Carter's (the shot-firer) body or disturbance of his clothes, in the space between the shot and the initial explosion, indicate not only the absence of disruptive forces, but show also that the gaseous movement was of low velocity. The same conditions were observed between the other local explosions.

A tram of coal had been left at the end of the level beyond explosion No. 6, before the disaster occurred. The tram was found neither damaged nor moved, nor was the coal disturbed (p. 20). It stood 2 feet 6 inches above the rails, and the blocks of coal were 15 to 18 inches higher than the wood-work; the top surface of the coal reaching a height of 4 feet. The hard faces of these blocks had been attacked by gases at a high temperature, causing them to burst out into globules of bituminous matter, which underwent further destructive distillation, leaving residues of comminuted coke. Some globules remained attached to the faces of the coal in different stages of production; but every ledge of coal, the edges of the wood-work of the tram, and the buffers, were loaded to overflow with accumulations of

coke, resembling small shot or coarse sand (p. 55). This comminuted coke had gravitated down to the resting-places where it was found, showing that there was an inadequate horizontal movement in the gaseous body to carry the particles beyond the tram; and admitting the element of time, necessary for the chemical changes in the theory already advanced.

At the *loci* of the local explosions, the trams were broken up and the coal was scattered. In explosion No. 2 a tram of coal weighing about 16 cwt., and standing about 8 yards from a door, was projected through the door as through a target, and fragments of tram, coal, and door, were found in a shapeless mass, some yards beyond (p. 16). Timber was broken down and splintered, arches were wrecked; a man who happened to be at the *locus* of one explosion was shockingly mutilated, and other men, a short distance from another explosion, were hurled away, and the clothes partly torn off their bodies (pp. 19–25). The condition of the roads, the materials, and the victims at the *loci* of the explosions, demand for their explanation forces moving at the velocity of an explosive wave.

In the spaces between the explosions, the roads were not damaged; the timber was not broken; the trams were uninjured and their contents undisturbed; the victims were not mutilated, and their clothes were undamaged. The phenomena in the explosions, and the phenomena in the intervening or antecedent spaces, therefore exhibited distinct modes of energy.

The *rationale* of these energies has already been suggested, but the phenomena at the Timsbury collieries throw further light upon the subject. When coal-dust distributed over the floor of a mine is subjected to heat, it is raised to a condition of intumescence, and the evolution of volatile matter is commenced; but if the temperature be inadequate, and the supplies of heat be not persistent, the distillation will be incomplete, leaving concrete residues containing unvolatilized tarry matter. If the coal-dust be subjected to a higher temperature in the same conditions of freedom and distribution, the volatile matter will be wholly extracted, and cohesion will be destroyed, leaving a residue of coke in the condition of granular powder. Both of these residues were found in the field of disaster; the former were produced from the floor to a height not exceeding 2 feet, the latter at a height of 3 feet to 4 feet above the floor.

The shot-like particles of coke at the tram exhibited perfect distillation; while the concrete residues disclosed only partial distillation of the coal. Both classes of coke were found in the same air-current within a few yards of each other; consequently there could not have been an appreciable variation in the periods of the distilling actions. Therefore, the perfect distillation at the top of the tram indicated a higher temperature than the imperfect distillation near the floor. The distillation at the top of the tram was necessarily effected by the heat in the educts from the coal-dust on the floor. When the educts rose from the floor they were at the temperature due to that stage of distillation which produced only partial volatilization; but when they reached the top of the tram they effected complete exhaustion of the volatile constituents, indicating the higher temperature due to the perfect stage of distillation. The educts had therefore gained an elevation of temperature at the height of 3 to 4 feet, and must themselves have been subjected to chemical changes. The nature of these changes has already been suggested, but the question may be further considered.

The coke with its hard, coarse structure, and the amorphous carbon fine and soft like flour, represent different states of the same substance, and must have been derived from different bodies. The coke is recognized as the residue of the destructive distillation of coal. The amorphous carbon is distinguished as the solid constituent of hydrocarbons, which in this case were the educts of the coal. The coke and the carbon were, therefore, produced from the coal and the educts respectively by distinct chemical actions following each other. The origin of the coke is perfectly well known; but the amorphous carbon may be produced by incomplete combustion of the educts, or by the oxidation of the hydrogen constituent, and dissociation. In the former case there would be a hydrocarbon flame, and having regard to the copious character and immense area of the deposited carbon, that flame must have been of great magnitude; and under its action the wooden materials and the bodies of the men and horses forming obstructions in the path of the flame, must have been burned and cindered. The surfaces of the wood were neither burned nor discolored; the men were burnt and blackened with coal-dust, but the skin and flesh was peeled, not consumed, and their hair and the hair of the

horses was simply singed; therefore, they had not been attacked with a hydrocarbon flame.

This flame must have yielded carbon monoxide and carbon dioxide as products, and the latter especially must have formed a considerable element in the atmosphere that filled the workings. Two survivors in the workings at the time of the disaster entered the stagnant atmosphere almost immediately after the occurrence, and rescue parties descended the shaft within an hour of the event. The circulation of the air was suspended by the destruction of the doors and collapse of the roads; the products of the chemical changes were, therefore, trapped and locked in between the intermittent falls, and could only be removed by diffusion. The rescue parties constantly advanced beyond the space in which diffusion was taking place, and entered the stagnant atmosphere to make apertures through the obstructions, with the object of rescuing the men either alive or dead. Favorable opportunities, therefore, occurred for testing for the presence of carbon dioxide, and its absence was shown by the candles burning brightly everywhere in the stagnant atmosphere; and by the parties suffering physiological sensations not known in an atmosphere containing carbon dioxide, and which were utterly inadequate for the effects that must have resulted had there been the quantity of carbon monoxide present that would have been produced by the combustion under notice.

Gaseous hydrocarbons undergoing incomplete combustion at the *locus* and at the moment of their evolution and separating out their carbon, could not bring about disruptive effects; and the numerous exhibitions of violence through the field of disaster are as inexplicable upon this hypothesis of combustion, as they are inconsistent with it.

The positive absence of the effects of flame, and of carbon monoxide and carbon dioxide, with the exhibitions of explosive violence, prove that the amorphous carbon was not obtained by the incomplete combustion of nascent hydrocarbons. The remaining source of this carbon was in the oxidation of the hydrogen constituent of the hydrocarbons, and by their dissociation.

The hydrocarbons were present, and an ignition-temperature. The absence of combustion, even incomplete, shows that there was inadequate atmospheric oxygen for the purpose. In these

circumstances the limited supply of oxygen would be seized by the hydrogen in the educts, and the hydrogen constituent of hydrocarbons, bringing about other changes already described, yielding the phenomena above referred to, and resulting in the separation of the carbon. The origin of the amorphous carbon is now traced to the gases of which it was a constituent, and which the coal-dust had yielded by distillation.

An illustration of this action is reported in the Ohio petroleum regions, where gaseous hydrocarbons are yielded by the bore-holes, and some portion is dealt with for the purpose of obtaining the carbon constituent which forms the subject under notice. These hydrocarbons are treated in limited supplies of air, and the hydrogen constituent only is oxidized, the carbon being separated in solid form, producing the article of commerce known as "Diamond Black."

Returning to the two classes of coke at the tram, with the results of this inquiry into the origin of the coke and the amorphous carbon, the observed elevation of temperature in the educts between the floor and the top of the tram, is found to have its explanation and confirmation in the presence of the amorphous carbon, which was produced by chemical changes in the educts with a great generation of heat.

The temperature of these changes reaches a point exceeding 2000° C.; therefore the distillation of the faces of blocks of coal and the perfect exhaustion of the volatile constituents will now be understood.

The suggestion advanced in *Coal-Dust an Explosive Agent* of chemical changes in the educts of the coal is now seen to derive further confirmation from the phenomena of the disaster at the Timsbury collieries, which verify the chemical actions that were propounded; and the theory recited in this paper (pages 115, 116), in explanation of the explosion at the Camerton collieries, is confirmed by the phenomena observed at the Timsbury collieries.

The identity of the phenomena of these two disasters is a fact, the importance of which cannot be overestimated; as they provide the origin and rationale of two explosions, not complicated by the presence of fire-damp, and which are found to have been brought about by gases derived from the coal. With this body of evidence, attention will now be directed to explo-

sions in mines which normally yield fire-damp, and an effort will be made to discover whether they were caused by that gas, which was supposed to have suddenly appeared in large volumes, or by the coal-dust always and everywhere present.

The Albion colliery-disaster in June, 1894, when 290 lives were lost, is examined on pp. 72–88. The records made of the calamity were of a general character, and were equally urged in support of opposing theories of its origin. The conflict of opinion was founded upon the fact, that materials in the same road had been violently displaced in opposite directions. The phenomenon was described by the respective advocates as "complex," and their explanation was that the "blast had passed over the ground twice" (p. 74). This hypothesis is examined at pp. 75, 76 and 77, and shown to be untenable; and the positions in which the materials were found, are proved to demand the exertion of explosive violence in the vicinity, with disruptive forces moving in all directions from a common center. The disturbances were found at intervals throughout the field of the disaster, and the terrible contrasts between the condition of the victims at these *loci*, and their condition in the intervening spaces, made it absolutely certain that there had been developments of explosive violence at intervals. The records, therefore, presented the phenomenon of intermittent local explosions.

The disaster had its origin in shot-firing, where timber was being removed by charges of gelatine-dynamite (pp. 83–85), and the surplus heat for distillatory action is estimated at about 250,000 gramme-units (p. 86). The coal yielded 15.19 per cent. of volatile matter (p. 81). In the immediate vicinity of this shot-firing, there were extensive residues of coked coal (p. 86), which is the evidence of the fact that a large quantity of heat was suddenly thrown into the coal-dust at this place, and that a series of regenerating chemical actions was established, identical with those originated at the shots in the Camerton and Timsbury collieries.

The road was heavily timbered; but with the exception of the frames displaced by the charges of gelatine-dynamite, there was no disturbance of the timber or strata for 90 yards upon either side of the shots; though neither timber nor strata could have offered an effective resistance to disruptive forces (p. 87).

At the end of the undisturbed space in each direction, disruptive effects upon the timber and strata were exhibited, indicating the *loci* of the initial gaseous explosions. This phenomenon of local disruptions, with antecedent lengths of roads in which there had been no disruptive force, was exhibited throughout the field of the disaster.

The disruptions occurred in large sectional areas, or where abnormal supplies of oxygen were available, and the activities terminated where the air was practically exhausted, and where a wet space opposed the progress.

Coke residues were extensively found in the workings (p. 81). At many points the blocks of coal upon the trams were in process of reduction into coke (p. 88), and amorphous carbon was suspended in the atmosphere, and in the effluent gases that rushed out of the shafts (p. 82).

It will be observed that there are fundamental identities in the phenomena of the disasters at the Albion, Camerton, and Timsbury collieries. They are each distinguished at their origin by gas-generating spaces, in which no disruptive energy was exerted, and are led by corresponding modes of development to exhibitions of explosive violence; so that each field exhibited numerous intermittent and local explosions. Residues of coherent coke, comminuted coke from the faces of blocks of coal, and amorphous carbon, disclose the correspondence of the chemical changes throughout the field of disaster, and indicate their identical regenerating action; and their terminations were brought about under the same conditions. These fundamental identities involve the important conclusion, that these disasters in gaseous and in non-gaseous mines, were brought about by identical series of chemical actions in the coal-dust.

At the Malago Vale colliery, Bristol, explosions occurred August 31, 1891, originated by the heat produced in the inexplosive ignition of a small quantity of fire-damp: and March 15, 1895, originated by the surplus heat of a charge of mining-powder. These are dealt with at pp. 88–92. They exhibited the principal phenomena under notice, and the important observation was made on both occasions of men lying either dead or unconscious in the atmosphere termed "after-damp," produced by the explosions, with a lamp still burning beside them (pp. 89, 92). The physiological sensations experienced in the

after-damp both by the rescuers and the rescued, were identical with the sensations observed in the stagnant air between the falls at the Camerton and Timsbury collieries (pp. 89, 92, 93).

At the Llanerch colliery, Monmouthshire, an explosion occurred in February, 1890, killing 176 men, which was originated by the heat generated in the inexplosive ignition of a small quantity of fire-damp (p. 93), which set up chemical actions in the coal-dust; and its phenomena were identical in every respect with the phenomena of the Albion and Malago Vale disasters.

The phenomena of the four explosions in mines normally yielding fire-damp, were therefore identical with the phenomena of the explosions in the Camerton and Timsbury collieries, which yielded no fire-damp.

The investigation is pursued by selecting great explosions in various districts throughout Great Britain, and examining their phenomena.

Evidence of the nature of the "after-damp" is given at pp. 98–101, showing that the atmosphere that filled the gaseous mines after the great explosions produced effects identical with those observed in the non-gaseous mines (Camerton and Timsbury), *i.e.*, the lamps burned brightly, and the same physiological sensations were experienced. The gases that brought about the explosions in gaseous and in non-gaseous mines must, therefore, have been identical, as the products of the chemical changes produced identical effects.

Attention is drawn to the fact that the exhibitions of violence required for their explanation the maximum explosive mixture of fire-damp, *i.e.*, an atmosphere containing about 10 per cent. of methane. One volume of methane yields in this explosive combustion, one volume of carbon dioxide, consequently carbon dioxide must form 10 per cent. of the atmosphere produced.

The most recent researches into the constitution of the atmosphere found in collieries and containing carbon dioxide (previously referred to, p. 111) are given at pp. 96–98. It is shown that candles are extinguished in an atmosphere in which carbon dioxide is present to the extent of 2.21 per cent., but that no loss of power is produced; and the first physiological sensation is felt when the quantity rises to 3.38 per cent., and that 7.32 per cent. produces violent respiratory distress. It is, therefore,

obvious that the after-damp which supported the flame of lamp and candle, did not contain one-fourth of the volume of carbon dioxide that is produced in the explosive combustion of fire-damp, consequently the explosions that produced this after-damp were not explosions of fire-damp. If confirmation were asked of this scientific induction, it could be afforded in the fact, that the physiological sensations experienced in the after-damp, are fundamentally different from the sensations felt in atmospheres containing carbon dioxide.

Other phenomena of the numerous calamities in gaseous mines are the presence of coked residues and amorphous carbon (pp. 103, 106). Their origin is characterized by undisturbed lengths of road (p. 107), at the ends of which the initial gaseous explosions occurred. The propagated explosions were at varying intervals, violent in their energy, local in their effects, exerting force in all directions (pp. 108–111); while the intervening spaces had known no violent energy. "Things that could easily have been moved were not disturbed" (p. 112). These local explosions were found in large sectional areas, or where the ventilation was concentrated by doors and obstructions (p. 113). The terminations of the explosive phenomena were in wet places or in impoverished air.

The phenomena of explosions in gaseous and in non-gaseous mines, therefore, have fundamental identities in subsidiary local explosions, separate in time and in space, with an identical series of chemical changes and products, that demand for their explanation an identical explosive agent; and the preceding remarks prove that agent to be coal-dust.

The mystery that has surrounded so many calamities in gaseous mines, must disappear with the knowledge that it is no longer necessary to invoke the hypothesis that a "blower," or an unknown accumulation of fire-damp was suddenly launched into the atmosphere of the mine, adequate in volume to form a highly explosive gaseous mixture in thousands of yards of road; as there was already at hand, in the coal-dust lying upon the roads, a practically unlimited supply of gaseous educts, capable of giving rise to explosion. These educts were made to flow into the atmosphere of the mine by processes which needed only the heat from an ordinary charge of explosive, or the combustion of a small quantity of fire-damp for their origina-

tion; and when once the activities were established at any local point, they carried death and destruction through the mine.

The theory advanced is, that the solution of the calamities that darken the history of coal-mining demands no assumption of the sudden and incredible accumulation of an explosive agent, but that the presence of coal-dust must give rise to the observed phenomena.

The conclusion of the Royal Commission upon Accidents in Mines, 1879, that coal-dust was not an explosive agent, which was founded upon the supposition that the great explosions in gaseous mines were caused by fire-damp, is now shown to be untenable (p. 120). The objection raised in the historical freedom of non-gaseous mines from explosions up to 1893, disappears with a knowledge of the conditions necessary to the chemical changes which produce an explosion, and which are shown to be only of recent completion (pp. 120, 121, 122).

The close relation between shot-firing and many colliery-explosions suggested the following remarks upon explosives:

The search for what has been termed a "flameless explosive," with the supposition that it would be necessarily used without danger of igniting gas or coal-dust, can only end in disappointment, as the danger in the use of explosives is seen to be the heat-energy developed in their combustion (pp. 123–125). Some experiments with explosives were made in 1893 by Mr. H. Hall (Inspector of Mines) for the "Royal Commission upon Explosions from Coal-Dust," and Mr. Hall's conclusions were given by Mr. E. E. Russell Tratman in the discussion upon Mr. Glenn's paper upon explosions, at the Institute meeting in February, 1894.* The Royal Commission adopted Mr. Hall's experiments, and reported that they believed some of the so-called flameless explosives were practically safe for all purposes, but that power should be given to prohibit the use of mining-powder except under stringent regulations (2d Report, pp. xxvii., xxviii). Mr. Hall's experiments are examined on pp. 125–128 of my book, and his conclusions that the use of mining-powder should be abolished in mines, and that ammonite and roburite should be substituted, are found to be unten-

* *Trans.*, xxiv., 900.

able. Mr. Hall's conclusion that roburite and ammonite "are incapable of igniting or exploding coal-dust," is also shown to be insecure.* He founds his conclusions upon the visible effects at the top of the shafts, and neglects the conditions at the cannon at the bottom of the shaft, where the heat was produced, and the phenomena originated. The charges of explosive varied, and the following quantities were used: 24 ounces of mining-powder, 10 ounces of ammonite and 8 ounces of roburite; but no attempt was made to determine the quantities of heat generated by these charges respectively, or the velocities of projection of the products of combustion from the cannon, or their spheres of action in the dust cloud. The element of danger in explosives was therefore not investigated or considered. The different behavior of the explosives in the experiments, is examined upon the pages named, where it is shown that Mr. Hall's conclusions rest upon a very partial basis of phenomena, and are not secure. Mining-powder is shown to have an element of safety not possessed by the "high explosives" (p. 125); and the precautions adopted in its employment are equally necessary when the other explosives are used (p. 128).

It is beyond question that if charges of mining-powder, roburite and ammonite be fired, as in ordinary mining operations, in corresponding positions, so that the products of combustion are thrown into coal-dust in the vicinity, and these products contain equal quantities of heat, identical effects in the coal-dust must follow.

The disquieting fact that the heat-energy in a charge of explosive can originate a devastating explosion in a mine, is divested of some of its terror by the knowledge that a simple application of water, thoroughly made, can easily effect the harmless surrender of that energy, and, consequently, the danger arising from coal-dust in the presence of shot-firing admits of a simple remedy (pp. 129, 130). In mines yielding fire-damp, precautions of an extensive character are necessary, as the combustion of a small quantity of fire-damp is found to produce sufficient heat to originate a disaster from coal-dust. In addition to adequate and effective ventilation, diluting and

* An explosion occurred on January 27 last, at Tylorstown Colliery, South Wales, causing the loss of 57 men, which was due to coal-dust and was originated by a shot charged with ammonite.

removing all emissions of gas, all open lights must be excluded and the approved safety-lamps adopted; the coal-dust must be removed as far as practicable, and an extensive and general system of watering kept in constant operation.

It is not possible, within the limits of a single paper, to offer more than a small portion of the evidence for the views I have here advanced; but I trust that the foregoing observations will be accepted as adequate to lead to a further study of the theory worked out in my books; and, in conclusion, I venture to hope that the *rationale* of a colliery-explosion, advanced in *Coal-Dust an Explosive Agent*, and developed in *The Origin and Rationale of Colliery-Explosions*, will help to give such a grasp of the nature of the danger to be guarded against as will strengthen the hands of all engaged in colliery-operations in contending with it, so that, in future, coal-supplies may be obtained without the terrible sacrifice of life which explosions have entailed.

Notes on the Walrand-Legēnisel Steel-Casting Process.

BY H. L. HOLLIS, CHICAGO, ILL.

(Pittsburgh Meeting, February, 1896.)

THE paper read by Mr. George J. Snelus, in 1894, before the Iron and Steel Institute of Great Britain* so fully and ably described the Walrand-Legénisel process that there remains very little to add beyond recording some changes in practice, made since that date, and observed by the writer last summer, at several continental works, and the further fact that the process has now been for some time in successful operation in this country.

Briefly stated, the Walrand-Legénisel process is an addition to the Bessemer operation, with the object of obtaining quiet and more fluid steel, and consequently sound castings and ingots. This is achieved by adding ferro-silicon at approximately the time of flame-drop in the ordinary Bessemer operation, and making an after-blow. The high calorific power of silicon is well known, and the rapid combustion of it immediately before

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* *Jour. I. and S. I.*, xlv., 1894, No. 1, p. 26.

pouring raises the temperature of the steel noticeably, and—what is likewise important—finishes the steel-making operation by the combustion of silicon to a solid (slag) instead of that of carbon to a gas. The consequent fluidity of the steel is well shown by the very small and intricate castings, running down to a fraction of a pound in weight, which are regularly made; but perhaps the best indication of the heat of the steel is shown by the fact that charges as small as 600 pounds have been blown successfully for several years at Paris, and often the charge has been poured entirely into small work, taking from fifteen to twenty-five minutes, without a sign of skull in the ladles. At the Potter & Hollis Foundry Company's plant, Chicago, heats of 1000 to 1200 pounds of steel have often been poured by means of two hand-ladles of 100 pounds capacity (necessitating the refilling of each ladle five or six times) and not less than twenty minutes taken for the casting, and yet the ladles have been entirely free from skull at the end.

Bearing in mind the calorific power of silicon and its well-known influence in the Bessemer process, one is tempted to inquire whether all this has not been done before, but in the investigation of the patent a most thorough search made through patent office reports, as well as metallurgical literature and proceedings of technical societies, here and abroad, failed to show anything in any way similar.

The details of practice vary considerably at the different works where this process is employed, but the features in common are briefly these: The vessels employed are the usual type of bottom-blown Bessemer converters. These receive the pig-iron in the ordinary way and the blowing is continued until about the time of flame-drop, when the vessel is turned down and ferro-silicon (containing from 10 to 12 per cent. of silicon) is added. An after-blow is then made, the time for terminating which varies at the different works. After this, ferro-manganese or spiegel is added to the vessel for recarburizing, and the steel is poured into large ladles on cranes, or small hand-ladles, according to the nature of the work. The blowing is generally controlled with the aid of the spectroscope. A mention of the differences in practice may be of interest.

At the works of Eugene Legénisel in Paris, there are two converters, one of 1200 pounds', and one of 600 pounds' capac-

ity. The sprues and heads and all steel scrap are melted with the pig-iron in the cupola and every cupola charge consists, after the first one or two heats, of about 80 per cent. of pig and 20 per cent. of steel scrap. The ferro-silicon and spiegel additions are melted in small and very ingeniously arranged cupolas, and are added to the converter by means of ladles. In blowing, the converter is turned down the first time just short of flame-drop, so as to leave enough carbon to act as an index in the spectrum of the after-blow. The silicon in the steel very seldom exceeds 0.05 and often runs down as low as 0.02 per cent. A very large proportion of the small work is cast in green sand.

At the Creuzot works of the Messrs. Schneider there is one 1500-pound converter. The steel scrap is added directly to the converter and the ferro-silicon and the ferro-manganese are heated red but not melted before using. The spectroscope is used to control the blowing, as at Paris; but the vessel is turned down earlier in the after-blow and the aim is to leave 0.20 per cent. of silicon in the steel.

At the works at Hagen, Westphalia, there are two 1200-pound converters. The blowing here is continued to the flame-drop for the first turning-down, and the after-blow is regulated more by time and the general appearance of the flame than by the spectroscope. The additions are melted in crucibles and the small work is poured from crucibles.

The Potter & Hollis Foundry Company, Chicago, has had for several months one 1500-pound converter in operation, and another is now being added. The steel scrap is added entirely to the cupola-mixture and the ferro-silicon and spiegel are added molten. The blowing is controlled practically as at Paris. The most important change in practice is in the blast-pressure; on the Continent very high pressures are employed—up to 30 pounds per square inch—while at Chicago not over 10 pounds has been used. Melting the additions in crucibles, as at Hagen, and in small cupolas, as at Paris, have been tried. Both ways are satisfactory and local conditions will determine which is more economical. The pin-bottom, as used at Hagen, and also the usual tuyere-bottom, have been tried; and it is perhaps too early to say which gives the better results.

The steel made by this process is satisfactory in every re-

spect. Entirely sound castings are obtained of any size, up to the capacity of the vessel, as well as ingots; and the steel shows high results under physical tests. Carbon, manganese and silicon are entirely under control; the first two being regulated by the added spiegel and the latter by the blowing. Phosphorus and sulphur depend, of course, on the stock used. Successive heats can be blown with practically identical compositions. At the Chicago plant the composition for the usual run of castings is from 0.25 to 0.30 per cent. carbon, between 0.10 and 0.20 per cent. silicon, and about 0.60 per cent. manganese. Such steel will give, unannealed, 75,000 to 80,000 pounds tensile strength, with 5 per cent. elongation in 8 inches, and annealed, 70,000 to 75,000 pounds tensile strength, with 12 per cent. elongation in 8 inches. The experience so often quoted, of obtaining less sound castings as the temperature of the steel rises, is entirely reversed, and it is found that the hotter the steel, the more solid and more satisfactory in every way are the castings.

A word in regard to the silicon in this steel: While it is possible to run the silicon down for regular practice as low as 0.05 per cent., it is found that nothing is gained by doing this, so far as the castings are concerned. By increasing the silicon (even to as high as 0.50 per cent.) the tensile strength is raised very much, without appreciably lowering the elongation.

Tests for magnetic permeability have been made with this steel, containing over 0.30 per cent. carbon and with manganese above 1 per cent., which have given the very best results. While it is not the intention of the writer to theorize, it may be remarked that the very high casting-temperature, together with the freedom from gas, seems to have resulted in physical conditions in the steel which modified the influence of the chemical constituents, as generally formulated.

The only trial of this process with large vessels that has come to the attention of the writer, consisted in a number of heats blown in 8-ton vessels at *Les Aciêries de France*, last summer. Here the object was to obtain soft steel for wire, and the resulting product showed less than 0.02 per cent. of silicon, 0.13 of carbon and 0.20 of manganese; the physical test running about 50,000 pounds tensile strength, with 27 per cent. elongation.

While it may be unwise to conjecture uses for this process in

advance of actual trials, it seems not improbable that where the increased solidity of the ingots justifies the small additional cost of their manufacture it will have a future in connection with the operation of large Bessemer plants. It has most certainly demonstrated its value in making steel castings.

To summarize what may be justly claimed for this process up to the present time:

1. It furnishes a steel foundry with exceedingly fluid, quiet steel in such quantities and at such intervals as are best adapted to the work to be poured.

2. It makes possible the successful casting of small and intricate pieces in steel.

3. Green-sand moulds can be used with satisfactory results; the steel being so fluid and holding its temperature so well that it can be poured slowly over a lip, and in this way not tear a green-sand mould and not carry in air, as is the case so often when bottom-pouring from large ladles is employed. It should be said, however, that when castings have to be machined for a perfect surface, it is better to use dry-sand moulds, for with green sand there is an unavoidable "pin-holing" at times in from the surface.

4. The relative small cost of installation, and the fact that a converter can be heated up in an hour ready for use, make it unnecessary to operate the plant continuously in order to obtain economical or satisfactory results.

It should be added that M. Walrand claims the same results in the basic process by substituting a high-phosphorus iron for the ferro-silicon addition, but with this the writer has had no experience.

The Embreville Estate, Tennessee.

BY GUY R. JOHNSON, EMBREVILLE, TENNESSEE.

(Pittsburgh Meeting, February, 1896.)

It is now generally acknowledged that the successful management of a modern foundry necessarily embraces a knowledge of chemistry, and especially a thorough acquaintance with the effect of the various metalloids on cast-iron, in order to

produce, with certainty and regularity, castings of the requisite strength and physical properties. Nothing so much assists in this work as having a strong pig-iron, low in metalloids, to act as a basis of mixture, and to carry cheaper grades.

In this connection, the writer has thought that a brief description of the Embreville property, and the iron made there, may be of interest to the Institute.

This property is situated in the extreme northeastern corner of Tennessee, on the banks of the Nolachucky river, one of the principal affluents of the Tennessee. It is in a region formerly dotted over with Catalan forges and charcoal blast-furnaces, the product of which was famous all over the South for toughness and ductility.

The present plant at Embreville is the successor of an old charcoal furnace and rolling-mill, erected some sixty years ago, which always turned out a high grade of iron, commanding a ready sale throughout the South.

As was the case with many of these early plants, the great expense of getting charcoal, the distance to market, and the crude methods of transportation, caused successive business embarassments, as a consequence of which the property passed through a number of hands. It was finally purchased in 1888 by an association of Englishmen, under the style of "The Embreeville Freehold, Land, Iron and Railway Company, Limited."

After considerable negotiation, this company induced the East Tennessee, Virginia and Georgia Railroad to build a branch from Johnson City, on that road, to Embreville, a distance of about eleven miles.

Immediately on securing the railroad, the company caused to be constructed by the Pittsburgh Iron and Steel Engineering Company a first-class blast-furnace plant, which was erected during the years 1891 and 1892. It consists of a stack 80 feet high by 19 feet bosh, furnished with three Cowper-Kennedy stoves, two McIntosh-Hemphill & Co. blowing-engines, pumps, boilers, stand-pipe, etc., the whole plant being well built, but of the Pittsburgh type, and unfitted for the ores of this section.

As was usual during the years from 1890 to 1893, a town-company was formed, and a good farm was spoiled by grading-operations.

The furnace was put in blast in 1892, and ran about twelve months in all, making an average of about 100 tons per day of fair iron, the lower grades predominating.

Owing to the collapse of the "boom" in 1893, the furnace was shut down, and stood idle until 1895, when it was bought by Mr. George B. Parker, of London, England, who promptly took steps toward putting the property in running order.

In October, 1895, all repairs and changes having been effected, the furnace was blown in, and has been running steadily ever since, turning out an average of from 80 to 90 tons per day, principally of high grades.

The output of the furnace seems small compared with its size and apparent capacity, but it is simply a question of quantity or quality. By driving rapidly, 150 tons per day could be made, but it would be principally of the three lower grades.

Such soft, brown iron-ore as the mines supplying this furnace produce cannot be driven rapidly through a furnace of this size, and yet produce a large proportion of foundry-iron. With ores of this kind, it is the writer's opinion (shared by several prominent members of the Institute) that the lining from the bell to the bosh-line should have a strong outward slope, *i.e.*, the interior of the lining, from bell to bosh-line, should have the shape of a truncated cone, with an outward slope of about 1½ inches per foot; and the bosh should be put as low in the furnace as possible in order to lower the melting-point, and should not be too large, these conditions necessitating a comparatively low furnace.

The difficulty in working a high furnace on a soft brown-ore burden is that, in the high furnaces, the ore melts too high up, and the iron coming down through the stock is more apt to be of lower grade, or colder, than in a furnace in which it melts lower down. By reason of this condition, the Embreville furnace, not having been altered from its original interior lines, has to be driven very slowly as compared with its theoretical capacity according to modern ideas (as based on Pittsburgh practice), the blast being increased only as the heat gets higher, which increase is instantly followed by a cooling of the whole furnace.

The iron thus far produced is, so far as the writer knows,

unique, the average contents of phosphorus in the whole blast so far having been only 0.12 per cent., and running in many cases down as low as 0.08 per cent.—certainly a remarkable figure for iron made from brown ore. Manganese runs from 0.40 to 0.50 per cent., and silica is easily controllable, the furnace having produced iron running all the way from basic open-hearth—both as to silicon and sulphur—to a high-silicon foundry, containing 5 per cent. of silicon and 0.008 of sulphur, with the fracture of No. 2 foundry.

The tensile strength of the iron is remarkable. The iron-breakers often throw the iron several times on the breaking-blocks before fracture, and the result of a number of tests at the Addiston Pipe Company's works gave it a tensile strength of 24,000 pounds, and a transverse breaking-strength of 2200 pounds.

Of course, an iron so low as this in phosphorus cannot but be red-short, and for that reason we have had considerable trouble in getting our customers to use it properly. If employed in too large a proportion for castings, it will chill, and the shrinkage will be very great. When properly mixed, however, it gives splendid results, and by reason of its great strength, freedom from impurities and scrap-carrying qualities, it is gradually finding its way to a class of customers who need a special iron of this nature.

Another interesting feature is the presence of lead in both ore and limestone. This element, though not present in proportions large enough to cause trouble by running out in quantity with the iron, gives considerable annoyance, owing to its curious property of causing the iron to part with its graphitic carbon, which lies in great masses in the runner after each cast. This throwing-off of graphitic carbon, in turn, gives the iron a "close-grained" look, which, of course, hurts it with the "rule-of-thumb foundry-man, who depends on the appearance of the fracture to tell him the nature of the iron. There is, also, a small quantity of zinc present.

The ore from which this iron is produced lies in one of the Tennessee limestone coves—Bumpass Cove—and is decidedly interesting from a geological stand-point.

According to Professor Lesley, a great fault took place just beyond the point where the Nolachucky river makes its last

break through the mountains into the great limestone valley, and this fault left, settled down behind the Potsdam sandstones and shales (known locally as the Chilhowee grit and shales), a syncline of No. II. limestone, which is overspread throughout a large part of its extent with from 6 inches to 30 feet of brown iron-ore.

From more recent examination, however, the writer is inclined to think that Professor Lesley was in error when he thought a fault made this deposit possible. A study of the geological features of the region reveals numerous short bends, and the writer thinks that Professor Lesley mistook an unusually short bend for a fault; in which opinion he has the support of the observers of the United States Geological Survey. The upper part of the bend has, of course, been denuded, leaving the sandstone and shale lying, apparently, over the limestone.

The property presents three varieties of ore, the first being a very siliceous specular iron-ore, containing Fe, 30; SiO_2, 50; and P, 0.5 per cent, lying high on the mountains, and found, of course, in the Potsdam measures. This ore lies in thin bands from 12 to 14 inches thick, separated by several feet of shale. There is an enormous quantity of it; but owing to its low iron-contents and high phosphorus, it is hardly probable that it will ever be profitable to work.

Descending the mountain some 200 feet vertically, a vein of massive brown hematite is reached, which is apparently very persistent, and is found on both sides of the cove. Some lumps of this ore have yielded 63 per cent. of iron, 2.5 per cent. of silicon, and of phosphorus, sulphur, and manganese only a trace; but the bulk of the ore is not nearly so good, averaging about 48 per cent. of iron.

Under the former management, this ore had not been touched, since it lay high on the mountain and was difficult of access. It is now being opened as rapidly as possible; but so little has been used as yet, that it is impossible to tell whether it will have any effect on the furnace-operation or not. It is hoped, however, that an extra amount of the "lump" will enable the furnace to carry a larger burden.

Still further down the mountain comes the limestone, which is overlain by the brown ore now used. As above observed, the depth of this ore varies from 6 inches to 30 feet, and occa-

sionally even more. In the limestone pockets, it has attained, in some instances, a depth of over 100 feet. The whole ground seems to be filled with particles of soft brown ore, so that it is usual with us to wash everything "from the roots down." The upper layer of red clay yields, on an average, about one car of washed ore to five of dirt, though often dropping to a proportion of one to ten; but when the ore-body itself is reached, the yield of ore is very much increased, running up in some cases as high as 60 per cent. of the material moved.

In the writer's opinion this ore has been broken up and washed down from the massive brown hematite lying above it and deposited in the limestone. This hypothesis is certainly supported very strongly by the configuration of the ground and the physical characteristics of the iron-ore, which yields a comparatively small amount of lump (about one-eighth) and runs 44 per cent. in iron and 10 per cent. in silica. It is soft and easily smelted, melting down very rapidly, with a tendency to clinker and to dirt-accretions on the walls. It also shows only a small quantity of phosphorus, but about 0.20 of manganese.

The washers employed are all double log-washers of the Thomas type with one exception, which has three logs.

The limestone underlying the ore is usually capped by a stiff black clay, locally called "ocher," which rarely contains any ore. The limestone is dolomitic, and works excellently in the furnace, containing only about 1½ per cent. of silica and producing a very fluid cinder. An average analysis would be SiO_2, 1.50; $CaCO_3$, 54; $MgCO_3$, 43 per cent.

Although a separate quarry is worked for this stone, yet, at the same time, in cases where limestone has to be blown out in order to get into the ore-body, it is broken up and sent down to the furnace, thus furnishing an exceedingly cheap flux. About three-fourths of a ton of stone per ton of iron made is required for this purpose.

At present the fuel used is Pocahontas coke, on which there is a 200-mile haul, partly over the Norfolk and Western and partly over the Southern railroad, a state of affairs naturally not conducive to cheapness of fuel. But, in case the coke of Big Stone Gap proves all that is anticipated, coke ought to be had at a reasonable rate.

The product of the Embreville furnace is marketed prin-

cipally in the west at foundries which need a strong, special iron. However, the iron is beginning to be in demand for malleable work. Possibly the malleable shops will eventually consume the larger part of it. The analysis of most of the iron thus far made is an ideal one for malleable work. The malleable-iron concerns which have bought small quantities of the iron for experimental purposes have in each case sent in return orders for larger quantities for further experiments, reporting their initial tests as entirely satisfactory. Pig suitable for malleable iron from south of the Ohio river is something of a novelty; but this development is in keeping with the other exceptional features of the Embreville iron.

The Effect of Additions of Titaniferous to Phosphoric Iron-Ores in the Blast-Furnace.

BY AUGUSTE J. ROSSI, NEW YORK CITY.

(Pittsburgh Meeting, February, 1896).

As is well known, practically all the phosphorus of the iron-ores smelted in the blast-furnace passes into the pig-metal, increasing its fluidity, but diminishing its strength to such an extent that, if phosphorus exceeds a certain amount, the metal is only fit for fine castings, not required to resist strains, and it cannot be used for the ordinary purposes of good foundry-iron.

No phosphorus being eliminated in the acid Bessemer process, the quantity admissible in pig-metal destined for that process must be very small, and hence the ores smelted must be strictly Bessemer ores, of which, in many districts, the supply is taxed to the utmost. On the other hand, very highly phosphoric pig, a metal of little or no value by itself, when refined by the Thomas-Gilchrist process in the basic converter, furnishes an excellent steel; but such pig-iron must be made from ores particularly high in phosphorus; and these ores are not everywhere abundant, and are much sought after for this special purpose.

But there are many iron-ores containing a moderate, though still important, amount of phosphorus, which yield a pig-metal

available neither for the acid nor the basic process of making steel, nor for economical use in the open-hearth, yet of indifferent value for foundry-purposes.

Titaniferous ores are almost invariably Bessemer ores; and the pig-metal obtained from them is a special iron, generally admitted to possess excellent properties. It is tough and hard, nearly white or light steel-gray in color, and characterized by its large percentage of combined, and small percentage of graphitic carbon.

It has occurred to me that by mixing in proper proportions titanic and phosphoric ores, the weakening influences of phosphorus on the metal might be counteracted to such an extent as to render such mixtures available for blast-furnace use, in order to obtain a fluid, and at the same time sufficiently strong foundry-iron for most purposes. If successful, this method would utilize a class of phosphoric ores not now possessing much value.

In experimenting in this direction, I have obtained some very curious and, to some extent, unexpected results, the communication of which is the special purpose of this paper. The experiments were numerous; they were repeated in many different conditions and proportions, with very good success; but for the present purpose I shall limit myself to a description of the four experiments comprising the most important points to which I desire to call attention.

In all these four cases the same furnace was used. The ores or ore-mixtures were intimately mixed with powdered charcoal and appropriate fluxes, so chosen as to exclude the idea of any phosphorus being contributed to the metal by their addition. They were placed in graphite crucibles, some 200 to 500 grammes of ores being used in one operation. The crucibles were charged in a furnace filled with charcoal, through which cold air was blown under a pressure of a few ounces.

The first two experiments, B, C, were made with phosphoric ores from the Sterling mines, N. Y., containing 1.75 per cent. of P_2O_5 (corresponding to 0.764 of phosphorus), and in round numbers 59 per cent. of metallic iron. In this ore the ratio of phosphorus to 100 of iron was 1.28.

Experiment B.—Sterling ore was used alone; the fluxes added were silica, alumina, lime and magnesia, practically

pure. Some powdered white ordinary glass (soda-glass) was added to promote fusion. We obtained a button (147 grammes) which, according to the analysis made (as were all those which follow) by the late Dr. Gideon Moore, of New York city, contained:

	Per cent.
Graphitic carbon,	2.99
Combined "	0.18
Total "	3.17
Phosphorus,	1.125

The iron was No. 2 gray, fairly good and strong.

Experiment C.—In this experiment the same Sterling ore was used, but it was mixed in the proportion of 3 Sterling to 1 of titaniferous ores from the Adirondacks, containing some 18 to 20 per cent. of titanic acid; phosphorus, 0.017; sulphur, 0.0236; and practically the same amount of iron as the Sterling (57.90 against 59 per cent.). Enough phosphate of lime was added to the mixture to bring the percentage of phosphorus up to that of Experiment B, that is, to a ratio of 1.28 P to 100 Fe. We obtained a button (147 grammes), to all appearances of the same grade (No. 2 gray) as in the case of the Sterling ore smelted alone, but it proved quite strong and tough, and required several blows of a heavy hammer to be broken on an anvil. It analyzed as follows:

	Per cent.
Graphitic carbon,	3.54
Combined "	0.33
Total "	3.87
Phosphorus,	1.229
Titanium,	0.35

The fluxes used were the same as in Experiment B, and the same quantity of powdered glass was added. Not only was the grade of iron not lowered by the addition of titaniferous ores, but the analysis proved that the button C contained much more total carbon, the excess being mostly in graphitic carbon. In preceding experiments with these same titaniferous ores, either in the blast-furnace or in the crucibles, under ordinary conditions, we had obtained white or rather light steel-gray iron of the following composition:

	Per cent.
Silica,	0.35
Combined carbon,	2.55
Graphitic "	0.23
Total "	2.78

And in another case:

	Per cent.
Silica,	0.29
Manganese,	0.34
Total carbon (mostly combined),	4.56

The two following experiments, D and E, were made with a phosphoric mixture prepared by adding to the same Sterling ore a sufficient amount of phosphate of lime to bring up the percentage of phosphorus to 2.045 (from actual analysis made). The mixture contained 57.83 per cent. of metallic iron; ratio of phosphorus to 100 iron, 3.53. The experiments were made simultaneously in crucibles placed side by side in the same furnace, exposed to the same firing for the same length of time, and removed at the same time.

Experiment D.—In this experiment the phosphoric Sterling ore was used alone with the phosphate of lime. The fluxes were the same as before, but no glass was added. The composition of the slag was approximately:

	Per cent.
Silica,	44
Alumina,	18
Lime,	28
Magnesia,	10

We obtained a button (72 grammes) of No. 2 gray iron, lighter if anything than button B (Sterling alone), of a fine close grain, but quite weak, breaking at the first blow. It contained 2.862 per cent. of phosphorus.

Experiment E.—In this experiment the mixture consisted of 80 grammes phosphoric Sterling ore (P, 2.045; Fe, 57.83) and 40 grammes of Adirondack ore, containing P, 0.023; Fe, 57.691, to which was added enough of phosphate of lime to bring up again the percentage of phosphorus in the mixture to 2.045, *i.e.*, to make the ratio of phosphorus, to 100 of iron, 3.53.

The fluxes used were exclusively silica, lime, magnesia, alumina, and a highly titaniferous slag which we had on hand, containing 38 per cent. of titanic acid, and some 14 per cent. of silica, with only lime, magnesia and alumina as bases. The resulting slag had an approximate composition of:

	Per cent.
Silica,	36
Titanic acid,	14
Alumina,	19
Lime,	23
Magnesia,	8
Phosphoric acid,	0.336

The button weighed 70.89 grammes. It was No. 1 Foundry, gray, large-grained, containing much graphitic carbon (3.98) and very little combined carbon. It showed on analysis 3.229 per cent. of phosphorus and 0.47 of titanium. It was strong enough, notwithstanding this abnormal amount of phosphorus, to stand several blows before breaking. In this case both the grade of the metal and the amount of graphitic carbon it contained were higher than in Experiment D and even higher than in Experiment C made with a less highly phosphoric mixture and a smaller proportion of titanic ore.

The curious fact brought out by these experiments is the peculiar influence which the presence of phosphorus seems to have exerted on the state of the carbon in the pig-metal obtained from mixtures of phosphoric and titanic ores.

From the titaniferous ores, used alone, a metal characterized by a large amount of combined carbon would have been normally expected, and, *a priori*, one would suppose that the addition of 25 per cent. of such ores to phosphoric ores yielding, when smelted alone, a No. 2 gray pig-metal, ought to lower the grade of the resulting iron. But the amount of graphitic carbon was greater in C (titanic) than in B (phosphoric) with a percentage of from 1.12 to 1.22 of phosphorus in the metal in both cases. Furthermore, we find that under the same conditions of treatment while a highly phosphoric ore used alone yielded a button of a No. 2 light gray iron, the same ore, mixed with 33⅓ per cent. of titanic ore yielded a metal containing the large amount of 3.89 per cent. of graphitic carbon, and strong, though containing 3.225 per cent. of phosphorus—an abnormal percentage, purposely chosen.

In short, while titanium in an iron-ore (at least under the conditions observed in smelting these ores hitherto) has a tendency to throw the carbon in the metal into the combined state, the presence of a considerable amount of phosphorus simultaneously with titanium, in a mixture, modifies the con-

ditions of existence of the carbon to the extent of throwing it almost all into the graphitic state, this effect being the greater the more phosphoric the mixture, and the greater the proportion of titaniferous ores used.

Standard Physical Tests for the Product of the Blast-Furnace, and their Value.

BY THOMAS D. WEST, SHARPSVILLE, PA.

(Pittsburgh Meeting, February, 1896.)

THE occasional reports of progressive furnace-men, giving the results of physical tests to prove the superior qualities of their pig-irons, have encouraged the writer to believe that the time is ripe for the discussion of such work, and the inauguration of some correct system by which the furnace-man and the founder could make intelligent comparison of the elements of strength, deflection, contraction, and chill presented in pig-metal.

In most cases the physical tests now made by furnace-men define but one quality in the iron, and are generally too vague to permit any true comparison. For example, one furnace-man tells us he can punch holes in his cast-iron, wishing it to be inferred that the iron is very soft and strong. Another furnace casts a long wedge, the thin end running to a knife's edge, which is supposed to prove the fluidity of the iron, for the founder's benefit. Another casts in square-cornered "chills," to define the hardness or softness of the metal. To the writer's knowledge these tests are not advocated by their originators as suitable for standards; and they are, to his mind, but evidence of unrest on the part of progressive furnace-men, who feel that something should be done in the line of physical tests.

The writer knows of no furnace-men who make and keep records of the strength of their iron obtained by means of tests made on recognized standard testing-scales; but he does know of a prominent furnace-salesman, who, not long ago, came into possession of a 1-inch square bar that stood 1400 pounds

between supports 12 inches apart, and who thought this result wonderful until he was informed that the writer had surpassed such a strength by 1100 pounds, with iron from the salesman's own furnace. Is there any article of common use, of which so much should be known and so little is known, as pig-iron? I doubt it. The selling of "pigs in a poke" can be greatly improved, to benefit both the furnace-man and founder.

It is true that we have some irons, such as ferro-silicon, of which physical tests would prove of little value to any one; but this is no reason why we should not advance in the direction of practical benefit to all concerned.

I believe all will admit that true progress in making and mixing cast-iron dates from the time the maker and user of pig-metal commenced the use of chemistry. All who have closely followed the history of chemical combined with physical tests for the past two years, will confess that they mutually aid each other, the former telling the constituents of iron, and the latter the effect which their combination produces.

The writer claims that progress in the science of either making or mixing iron requires the study of the physical as well as the chemical properties.

Two years ago, when few furnaces making foundry-iron, or foundries using it, were utilizing chemistry, the writer predicted it would not be long ere the furnaces, at least, would all have laboratories. This has largely come to pass; and the writer believes it will not be long before all will take the same interest in the question of physical tests as is now taken in chemical ones. There is certainly a great deal to be learned by combining these two lines of inquiry. Many erroneous ideas now entertained will be corrected; and the furnace-man will often find a knowledge of the physical properties of his iron before it left his yard, a means of self-protection, in enabling him to lay the blame of subsequent failure at the founder's door.

It is not the purpose of this paper to discuss all the qualities to be investigated; but to show that there are such qualities, the writer will call attention to one in particular.

It is generally known that low shrinkage and contraction are excellent qualities in pig-iron; and we often hear Lake Superior ores spoken of as making "high-shrinkage iron." Why should they be different from any other ore in this respect?

Why, of two irons giving the same chemical analysis, should one show greater shrinkage and contraction than the other—if such is the fact?

These are questions surely worthy of research, and not to be investigated without physical tests. All experienced in iron-work will admit that it would greatly enhance the value of castings in steel or iron, and make it practicable to produce durable castings that cannot be attempted at present, could cast-iron have its greatest evils, shrinkage and contraction, decreased. Who can tell what may not be discovered in this line by furnace-men, installing physical tests in their laboratories? There is a difference of opinion as to whether chemical analysis will permit physical qualities to be truly estimated. I do not wish at present to commit myself positively on this point; but I will say that, by reason of discoveries which I have made in conditions which are controllable in changing the physical character of iron, my inclination is to believe that chemistry will permit the physical properties to be closely estimated, when we advance sufficiently to possess true knowledge of the conditions which control the physical effects.

I know of no better place to obtain by experiment a knowledge of the numerous elements affecting the physical properties of cast-iron than right where the iron is made.

I cannot conceive of any one disputing the value of physical tests in the laboratory connected with a blast-furnace; and in accordance with this conviction, I respectfully present, for the consideration of all interested in this work, a few ideas and plans which I believe to be based upon correct principles, and able to stand the test of criticism and experiment. My suggestions and methods are offered freely to any who may wish to adopt them.

The importance of a correct standard system for such tests, as making comparison possible between different furnaces, or the same furnace at different times, is self-evident. The records of standard tests are always comprehensible and available for reference long after their immediate occasions have passed; whereas irregular tests tell an imperfect story to begin with, and tell it for one occasion only. If strength-tests are made on a 2-inch bar at one place, on a 1 by 2-inch bar in another, etc., or if contraction is taken from bars differing in sizes from those

of which the manager has any record, his past experience and records will be of very little value for reference or comparison, and he must commence anew to build up records, renew his studies, and lose much time before he can get to the point at which he would have started, had he been able to utilize his own past records and those of others.

Money and trouble are sure to be saved to the owners of furnaces and foundries by the adoption of standard tests and the keeping of complete records thereof; because every concern must at some time or other install a new manager, and serious losses are often incurred before a new man gets a good hold of all the reins. It is to the interest of all owners of plants, to aid him as far as possible, by furnishing him with intelligible records of past research and experience.

Finally, it is only when the facts to be discussed are stated in a form understood by all, and permitting their proper comparison and the estimation of their relative pertinence and weight, that scientific debate in technical journals or associations is practicable and most profitable.

The advisability and value of a standard are thus clear. Whether that proposed in this paper be acceptable or not, the writer is deeply interested in the establishment of some correct, convenient, and practicable method, and would do his part to inaugurate any other system that could be proved to possess these features.

It is well known that the writer is an advocate of the round test-bar, cast on end, and has published several illustrated papers discussing that form.* The statements and illustrations of

* To avoid the necessity of repeated citations, the following list of the writer's former papers is here given, for the use of those who may desire to study more fully the views suggested in the present paper:

"The Erratic Square Test-Bar," read before the Foundrymen's Association, at Philadelphia, May 3, 1894, and published in the *Iron Age* and in the *Iron Trade Review* of May 10, 1894.

"Round *vs.* Square Test-Bars, and the Utility of Transverse, Tensile, Crushing and Impact-Tests," read before the Western Foundrymen's Association, at Chicago, May 16, 1894, and published in the *Iron Age* and in the *Iron Trade Review* of May 24, 1894.

"Comparisons of Strength in Specialty-Mixtures of Cast-Iron," read before the Western Foundrymen's Association, October 18, 1894, and published in the *Iron Age* and in the *Iron Trade Review* of November 1, 1894.

"Specific Gravity and Physical Tests of Cast-Iron," read before the Iron and

this paper are, however, new, though partly in line with what he has presented elsewhere on the subject of testing; and the suggestions here made will be found of value, not only for the furnace-man, but also for the founder, and all persons interested in the subject of cast-iron.

The first point I wish to mention, is the value of re-melting samples of the furnace-casts. The occasional re-melting of samples of casts, in a small cupola, cannot but aid the advancement of research, and serve as a check on chemical analysis, and often as a protection to the furnace-man, by enabling him to learn what the founder can do in changing the character of iron after it has left the furnace-yard. A little cupola will also often be convenient for casting small pieces for repairs that may be needed between the furnace-casts, or when a furnace is out of blast.

A furnace-man is often not informed of complaints concerning his iron until it has been all melted up; and then he has generally no remedy other than to inspect the castings claimed to have been made from the iron complained of. As a founder, I know there are ways in which the original character of pig-metal can be so altered in the cupola as to place upon the furnace-man the blame for bad results for which he is not justly responsible. In such cases, the re-melting of a sample by him

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Steel Institute at Birmingham, England, August 20, 1895, and published in the *Iron Trade Review*, September 5, 1895; also remarks in discussion of the subject before the American Society of Mechanical Engineers, in December, 1894, published in vol. xvi. (1895) of the *Transactions* of that society, p. 571.

"Contraction *vs.* Strength of Cast-Iron," read before the Foundrymen's Association, September 4, 1895, and published in the *Iron Age* and in the *Iron Trade Review* of September 12, 1895.

"Stretching Cast-Iron, and Elements Involved in its Contraction," read before the Western Foundrymen's Association, November 20, 1895, and published in the *Iron Trade Review*, November 28 1895.

"Segregation of Iron at the Furnace and Foundry," read before the Foundrymen's Association, December 4, 1895, and published in the *Iron Age* and in the *Iron Trade Review*, December 12, 1895.

Also the following articles:

"Plans and Methods Necessary for Obtaining Comparative Physical Tests for Cast-Iron," published in the *American Machinist*, August 30, 1894.

"Notes on Relative Tests for Cast-Iron and Methods of Testing," published in the *American Machinist*, November 1, 1894.

"Jugglery in Testing Cast-Iron," published in the *Iron Trade Review*, March 28, 1895.

might at once exonerate him. The expense of a small sample-cupola need not frighten any furnace-man; he can erect one for $20. In fact, the writer erected one, which went into blast January 17, 1896, which did not cost $6, and took but seven hours' labor of one man from the time ground was broken until the bottom- and top-filler were at work. A cast was made in ten minutes after the iron was charged. This cupola was made of an old shell, 12 inches in diameter and 30 inches long, which was being kicked around our foundry-yard. It had been used a few years back, in an industrial street-parade, for casting horse-shoes, which were thrown to the people as the wagon rolled along, the blast being furnished by means of an old pair of hand-bellows. If iron could be melted under such conditions, in such a baby-cupola, no one need hesitate to believe that it could be conveniently done in a small cupola at a blast-furnace, where all the blast required could be steadily supplied.

The following record gives chemical and physical tests of a furnace-cast, taken January, 18, 1896, at the Spearman furnace, Sharpsville, Pa., and presents one form for such records:

Physical Tests of Furnace-Iron Taken January 18, 1896.

No. of Test.	Contraction.	Deflection.	Strength.	Fluidity.	Chill.	Diameter of Bar.	Strength per sq. in.
	Inch.	Inch.	Pounds.	Inches.	Not taken.	Inch.	Pounds.
L	$\frac{6}{64}$	0.12	2[illegible]00	$4\frac{1}{2}$		1.194	2054

Physical Tests of Cupola-Iron.

No. of Test.	Contraction.	Deflection.	Strength.	Fluidity.	Chill.	Diameter of Bar.	Strength per sq. in.
	Inch.	Inch.	Pounds.	Inches.	Not taken.	Inch.	Pounds.
2	$\frac{5}{64}$	0.08	2220	5		1.242	1907

Analysis of Furnace-Iron. *Analysis of Cupola-Iron.*

Silicon.	Sulphur.	Silicon.	Sulphur.
Per cent.	Per cent.	Per cent.	Per cent.
1.02	0.034	0.81	0.056

NOTE.—The number of inches given under "fluidity" in this record is directly measured on the fluidity strip, seen at S in Figs. 9 and 12.

The recent advance of chemistry in the art of founding, and the development of practical methods to assist accuracy in the grading and mixing of pig-metal, have greatly increased the certainty of obtaining desired results in castings over that which existed two years ago.

The writer claims, that, if the furnace-man furnishes a correct analysis of a well-mixed iron, which has been checked by physical tests at the furnace, and the founder or user accepts the same, the furnace should be relieved of further responsibility.

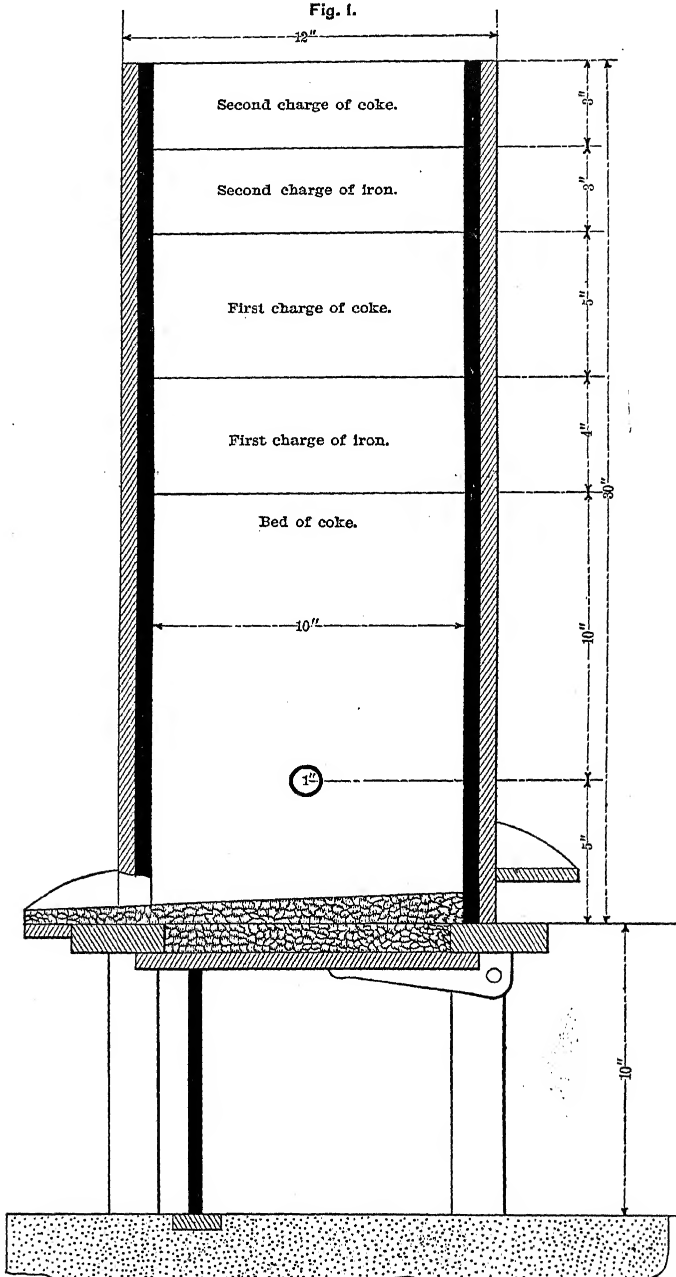
The day is past for tolerating the blind, ignorant practice which we foundrymen followed, a few years back, in mixing iron. The wonder is that we ever "hit" what we wanted, when we consider how deceptive is the fracture of pig-metal as guide to its true "grade." I am aware that only about one-tenth of our present founders have kept up with the progress of utilizing chemistry in mixing their iron; nevertheless, I say, when the furnace-man has done his part, let the founder study to do his by calling chemistry to his aid, or else get out of the business and stop his growling about "bad iron."

There is no "bad iron." All can be utilized in some class of work or other. All that is wanted is a knowledge of its chemical and physical properties; and when the furnace-man and founder understand these as they should do, pig-iron or any grade or quality need never be shipped in the wrong direction. It is simply a question of "carding the car" right, to have a furnace-man clean his yards, and have no complaint about his iron, however "bad" he may occasionally make it.

The foundry-iron of the above analysis is an excellent grade to make a machinable, strong casting for very heavy work, such as should not be under 3 inches thick in its lightest part, if all pig be used; but if the furnace-man gets the wrong shipping-card on the car, and some unprogressive founder receives the iron, and because it may look "soft" or "open-grained," tries to mix one-third scrap with it, for light or medium castings, he abuses the furnace-man because his castings crack and come out "white iron."

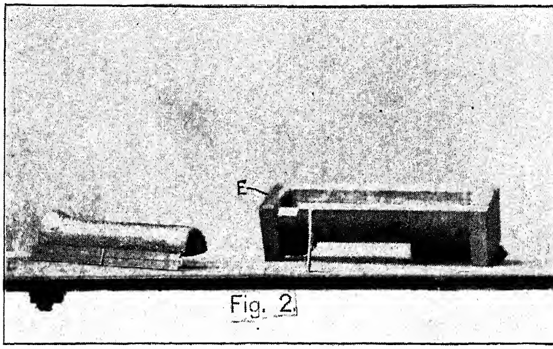
The cupola illustrated in Fig. 1 is the smallest I know of, now used for practical purposes. Before taking a "heat" out

Fig. 1.



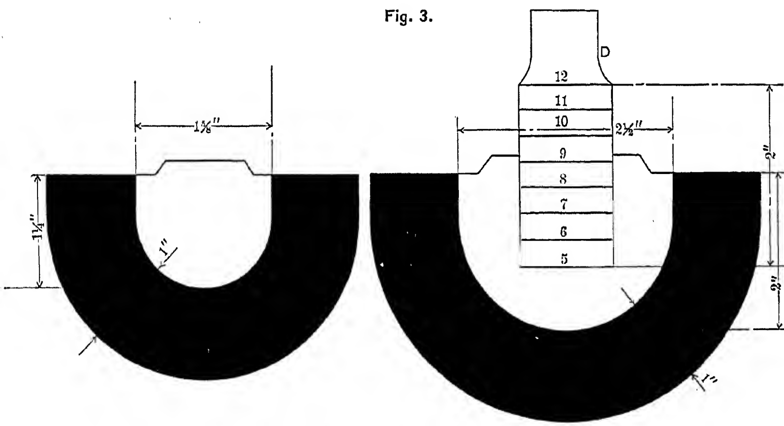
BABY-CUPOLA.

Fig. 2.



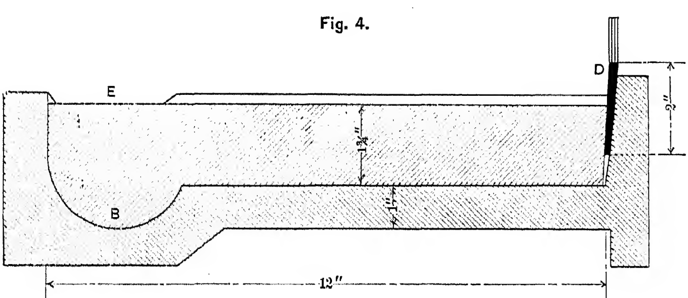
Iron Chill-mould, and Specimen for Contraction-test.

Fig. 3.



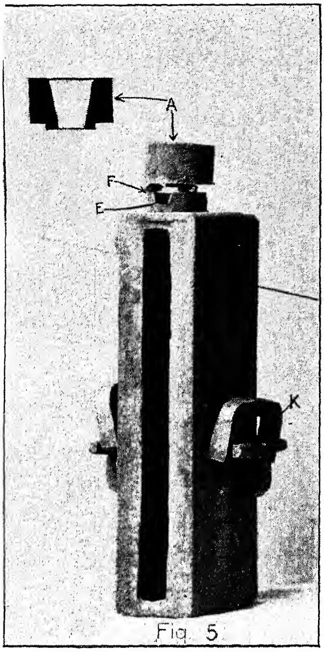
Cross-sections of Chill-moulds.

Fig. 4.



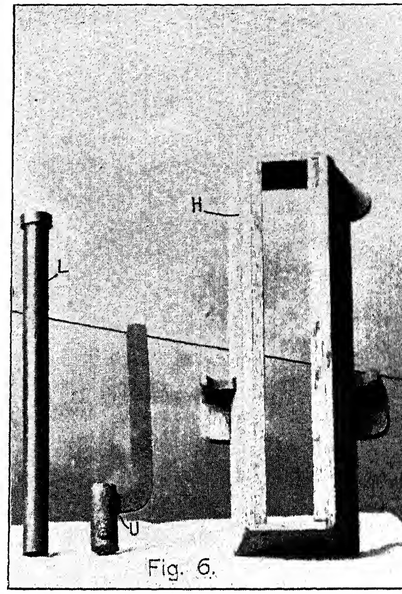
Longitudinal Section of Chill-mould.

Fig. 5.



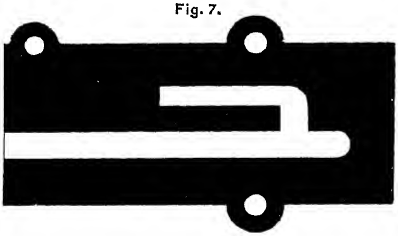
Mould Ready for Casting.

Fig. 6.



Flask and Bar.

Fig. 7.



Plan of Mould-board.

Fig. 8.

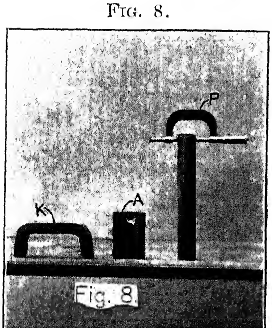


Fig. 9

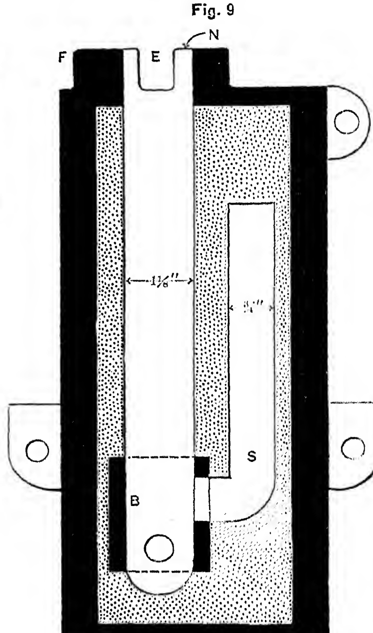
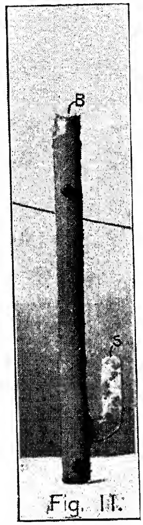
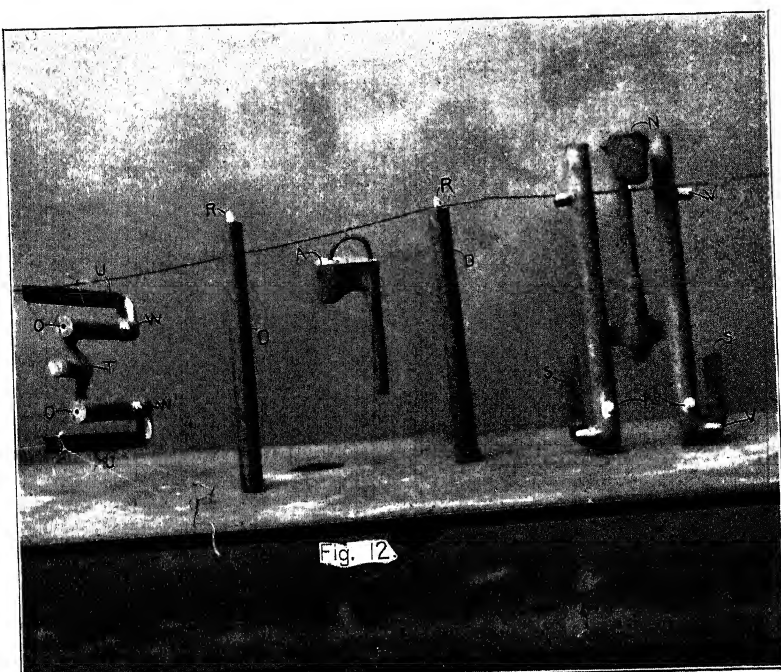


Fig. 11.



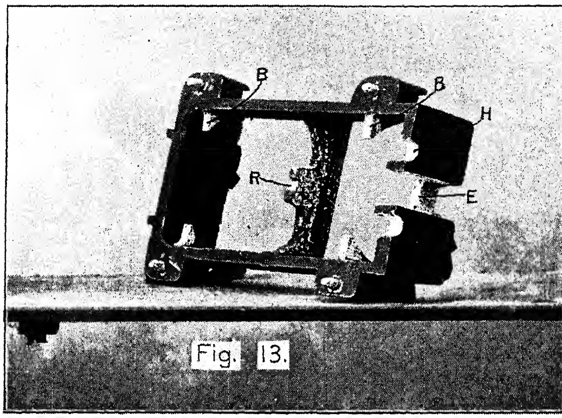
Test-bar with Fluidity-strip.

Fig. 12.



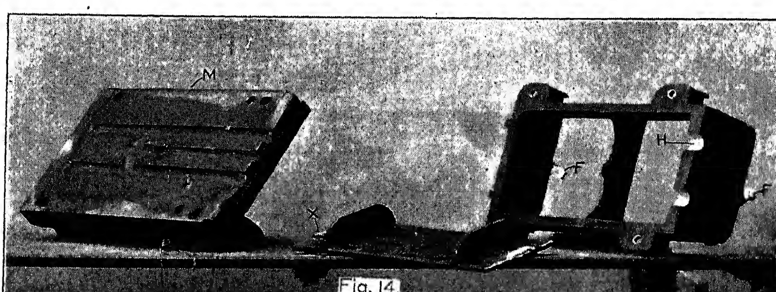
Double Test bars, Patterns and Casting.

Fig. 13.



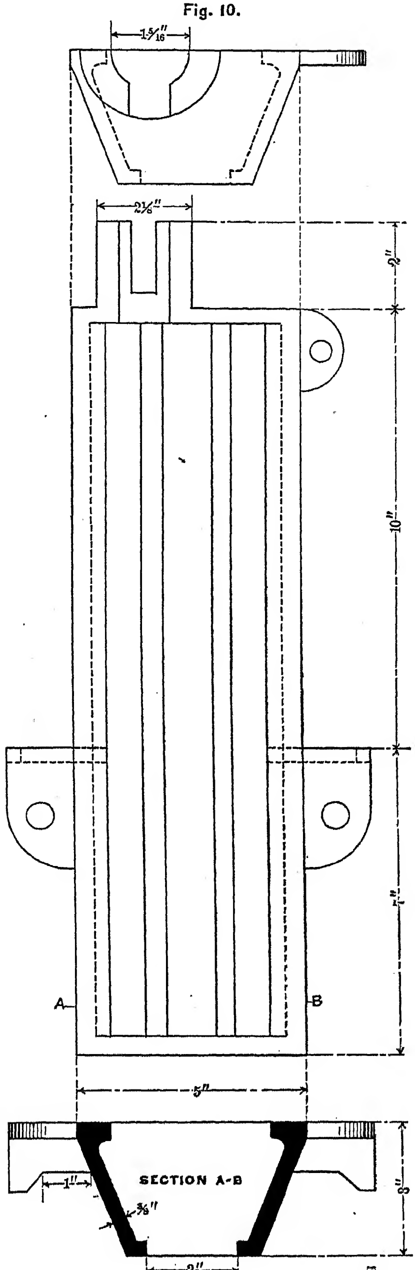
Nowel Flask for Double Test-bars.

Fig. 14.



Mould-board, Bottom-plate and Cope for Double Test-bars.

Fig. 10.



Single Test-bar Flask.

of this baby-cupola, there was but one point that I felt doubtful about in practice with such a small size for the work I intended it to perform, and that was, whether it would increase the sulphur, by re-melting, more or less than is done on an average in the large cupolas commonly used.

Owing to records kept at our foundry for the past three years of the analyses of the pig-metal that goes to make exacting work (in which only shop-scrap can be utilized), and of the castings produced, we are enabled to judge fairly of the increase of sulphur by re-melting. I am pleased to say that the increase in sulphur caused by re-melting in the baby-cupola cannot be regarded as any higher than would result from re-melting in large cupolas. If anything, it is a little below what might be expected with fair coke and such iron. This, I would say, is due to the iron not remaining in the baby-cupola as long as in ordinary foundry-cupolas.

The longer iron remains in a semi-molten state incased in coke, the more will its sulphur be increased; while, of course, the higher the sulphur in the fuel, the more is absorbed by the iron in re-melting.

The reduction of silicon in re-melting, is the greater, the higher the melting-temperature in the cupola, and the longer the metal is subjected to this temperature.

I will now proceed to describe a system of testing which I installed at a furnace in Sharpsville, January 17, 1896, in which the managers took great interest, and which they are using with much profit to themselves.

The outfit includes one Olsen transverse testing-machine of standard make, one cupola, two flasks, and chill pig-moulds with a test-bar pattern and mould-board. An excellent feature of the whole equipment is that it need not cost over $100, including the testing-machine, which is from one of the best builders in the country. The price of such an outfit is no more than a furnace might have to pay for freight on one car of condemned iron.

The Cupola.—Fig. 1 shows the cupola used. It may have a "drop bottom," as shown, or it may simply rest upon a plain plate, and be tipped by hand to clean it out, after the conclusion of heats. The figure itself explains all details necessary to the construction and charging of the cupola.

The blast used is cold, so as to be the same as in foundry-practice. It may require a few trials to find out what pressure of blast will give the best results. It should not exceed 10-ounces pressure at the cupola, and will be found generally to work best at from 6 to 8 ounces, where two one-inch tuyeres are used. With larger tuyere-area, less pressure will be required.

The cupola should have its bed of coke well on fire before the iron is charged, and the latter should be distributed evenly all over the surface of the bed, the largest pieces being placed in the middle.

I have melted one-quarter of a common-sized pig all down in fifteen minutes from the time it was charged. This is mentioned merely to show that the baby-cupola can deal very rapidly with large chunks of iron.

The melted iron should be held in the cupola until one charge is thought to have been all melted down, before it is tapped out. A charge of iron may range from 20 to 50 pounds; and several charges may follow, having a layer of coke between them, from 4 to 5 inches in thickness. For a heat over thirty minutes long, some readily-fused flux may be advantageously used to make a thin slag, which could be run off at the tap-hole or at a slag-hole, provided for the purpose, about 2 inches above the level of the tap-hole. If but one charge of iron is being melted, let the lowest pressure of blast found permissible with utility be left on, up to the time that about 2 pounds of melted iron runs out of the open tap-hole. After this flowing of metal, plug up the hole and increase the blast-pressure a few ounces, so as to bring down the iron quick, and collect it in a good body, which will maintain its fluidity while it remains on the bottom-bed before being tapped. In letting out the fluid metal, make a large hole and have a warm ladle to receive the liquid iron. In pouring a test-specimen, be sure the ladle is well skimmed, to prevent any dirt, scum or oxide from passing into the mould.

The lining used for the cupola is simply a coating of fire-clay from $\frac{3}{4}$ to 1 inch thick. It could, of course, be lined with fire-brick; the diameter of the shell being proportionately increased.

The baby-cupola shown is one which experimenters and college instructors could well use for giving instruction in

melting, and will be of value for scientific research in all cases where the melting of small masses will answer all practical purposes.

Horizontal Chill-Moulds for Measuring Contraction.—In explaining the operation of casting the test-specimens, attention is called first to Fig. 2, which shows a perspective view of the iron chill used, and the specimen obtained therefrom for testing contraction. Two sizes of these contraction pig-moulds can be used, or only one, as the furnace-man may deem best, in following out experiments and tests, as described later on. Fig. 3 shows cross-sections through the middle of the respective iron moulds; and the larger cross-section shows also the tapering-rule, D, applied at the end of the mould, to measure contraction. It will be noticed that the thickness of these miniature pig-moulds or chills is 1 inch. Any variation from this thickness would affect the depth of the chill. It is, therefore, necessary that care should be exercised to have always the same thickness in any standard chill pig-mould which might be adopted. The writer does not wish to be understood as advising records to be taken of the chill from the test-specimens, except in cases where very fine results are desired; and then note must be taken of the fluidity of the metal at the moment the chill-specimens are poured. This is done in the writer's system by means of fluidity-strips attached to test-bars, as at S, in Figs. 9 and 11, and also in Fig. 12. These strips are $\frac{1}{8}$-inch thick at the base, and 10 inches long, running up to a knife-edge, as arranged on the pattern shown at U, Figs. 6 and 12.

The writer has clearly shown in other papers how sensitive a chill is to the effect of any change in the fluidity or temperature of the molten metal, and the necessity of noting these conditions before any value can be attached to chill-records. The hotter iron is poured, the deeper the chill.

In Fig. 9 a chill-piece will be seen at B, which is the same as shown at A, Fig. 8, and is $\frac{1}{2}$-inch thick by 3 inches long, and made of soft steel. Only one side or half of the test-bar is here considered in measuring a chill for record. For iron above 1.25 per cent. silicon and no higher than 0.05 per cent. in sulphur, the system of obtaining chill-records indicated in Fig. 9 will work very satisfactorily. For iron lower in silicon or higher

in sulphur, it may be often necessary to have a larger body of iron, in order to prevent a specimen being chilled all the way through. In such cases, chill-blocks, as shown in Figs. 2, 3 and 4, may be required to obtain chill-records. But if any value is to be attributed to the chill-records, the fluidity should be tested at the same time, by the means shown in Fig. 9.

Fig. 4 shows a longitudinal section through the chill pig-mould of Fig. 2. The well at B is provided to prevent cutting the mould in pouring, and to cause the bar to pull towards one end in contracting, so as to permit the contraction to be readily measured by means of the tapering rule, shown at D. This test-specimen, being 12 inches long, provides a convenient length for measuring the contraction, and can be also readily broken to note its fracture, or can be drilled to obtain samples for analysis.

The sections in Fig. 3 show that the bottom-surface of the chill-mould is round, possessing no corners to cause any one part of the specimen to be chilled deeper than another (thereby causing internal strains and preventing natural contraction of the iron, owing to one part of the specimen being thrown into higher combined carbon than another). This consideration, the writer believes, will cause any one making a study of the subject to agree with him in advocating the principle of the round chill.

The tapering rule, D, Figs. 3 and 4, is graduated on one side, as shown, to measure the contraction in the sixty-fourths of an inch. The rule is cut off on the small end at a point where it is $\frac{1}{16}$-inch in thickness. From this the taper runs up 2 inches, at which point it measures $\frac{3}{16}$-inch. The distance between the $\frac{1}{16}$- and $\frac{3}{16}$-points is then equally divided by six lines, as shown, so as to read to the $\frac{1}{64}$ part of an inch, according as the space of contraction will permit the rule to be inserted between the chill-mould and the pig-specimen, as shown. The lines being $\frac{1}{4}$-inch apart, the scale can be easily read; but the rule could, of course, be graduated finer if desired.

The study of the element of contraction, as it can be defined from the pig-specimens, will prove very valuable, and, in time, may enable a tester to know at a glance, without further research, the true grade of an iron. It may eventually be a means to detect deception, which is now known to exist in the

fracture of direct metal, and also to learn the true effects of re-melting iron, and which metalloids effect the greatest contraction in the iron.

At E, in Figs. 2 and 4, will be seen a depression of about ¼ inch below the top-surface of the chill-mould. This is to provide means for a "flow-off," to insure the chill-specimens being always of the same thickness and prevent any iron running over the edges of the mould to retard free contraction in any manner. The chill-mould, of course, is set level.

By using together the chill-moulds of both sizes, as shown in Fig. 3, an excellent illustration will be afforded of the reasons why many castings crack or pull apart, owing to the work being badly proportioned. The small pig test-specimen will always show a greater contraction than the large one. Such ill results in cracks, etc., are often placed on the furnace-man's shoulders by claiming that he had sent "bad iron." Should a furnace-man not care to use these two sizes of chill-mould at one time, he may, under proper conditions, adopt either for constant use. In the case of very low grades of iron it might be necessary to adopt the larger chill-mould, since in the smaller one the iron might "go all white."

Test-Bars Cast on End.—In moulding test-bars for determining transverse or tensile strength or the deflection or stretch of an iron, the writer has devised a very simple design of a flask and one which would not require a $4-per-day moulder to make the mould. Any intelligent laborer can be taught in a very few minutes how to mould and cast such bars successfully; and it can be easily done in about two minutes.

In starting to mould a test-bar, the round test-bar pattern, L, and the fluidity-strip pattern, U, Fig. 6, are laid in the recesses of the mould-board, Fig. 7, which has been previously solidly placed. The half-flask, H, Fig. 6, is then laid on the mould-board, rammed up and rolled over, and then the cope is put on; clamps, as at K, Figs. 5 and 8, having been put on to hold the two parts close together while the cope is being rammed up. Before lifting the cope, the test-bar pattern, L, is pulled out endwise. The cope is now lifted off; the fluidity-strip pattern, U, is drawn out; the cope is put on and clamped; and the mould is up-ended ready for casting, as seen in Fig. 5. The iron cup, A, Fig. 5, is used for the purpose of providing a wide funnel

to pour into and keep the dirt from passing down with the iron. The slot cut in the iron end of the flask, as seen at E, Figs. 5 and 9, is to prevent the iron, as the mould fills up, from rising high enough to touch the under-side of the cup. Should the metal in coming up quickly, as it does, strike the under-part of this cup, an explosion would occur, to make the iron fly in all directions. By the plan devised such accidents are prevented.

In cases where the fluidity- and chill-tests are not desired, and a plain round test-bar only is wanted (which, for general furnace-purposes, will serve many ends), a plain round pattern, as at L, Fig. 6, which in the rough is 1⅛ inches in diameter, or in fine figures 1.1284 inches, that is, equal in area to a 1-inch square bar, is all that is required. It is well to have the lower end of this pattern made a little pointed for about ⅜-inch of its length, so as not to give a flat sand surface for iron to drop on, to "cut the mould," as in the case where the bar is entirely square on the end. In making this strictly plain, straight, round bar, the cope need not be lifted off, as the pattern can be pulled out endwise and the flask immediately up-ended, ready for casting, in less time than it takes to tell it.

Some might think a pattern rammed up on end in a wooden box would answer just as well. To do this and not have any swells on the bar requires considerable care in ramming the mould. By the plan here presented, no more time is required, and there is every assurance of obtaining a perfect, even, true round bar, free of all swells for its entire length, and without a joint-mark or flaw in it. These are essential requirements for a test-bar.

Should it be desired to cast but plain bars, without the attached fluidity-strips, the hole in the end of the flask, as at N, Fig. 9, could be placed in the center of the flask instead of where it is shown in the figure.

Fig. 10 is a sketch showing all the dimensions of the single test-bar flask shown in Figs. 5 and 6.

In moulding the bars, have the sand the same "temper" at all times, as far as such is possible, and endeavor to have the ramming done to the same degree of hardness. The result should be sufficiently soft to permit the iron to lie "kindly" to the mould, without bubbling or blowing, as the metal rises.

At the same time, the degree of hardness from ramming should be such as not to permit the head-pressure in casting to increase the lower end of the bar more than $\frac{1}{16}$-inch above the size of the pattern. A uniform temperature and fluidity in the metal before casting is highly important, and the founder should be educated to judge of this by the eye.

The micrometer seen at P, Fig. 8, is shown as applied to the measurement of a bar at its point of fracture, for computing strength per square inch, when fine results are desired. This is important in obtaining strength-records, as the least variation in the measured diameter of a bar can greatly affect its recorded strength.

Fig. 11 shows a single bar with its fluidity-strip, S, as taken from a mould. The two projections shown on the bar in this figure are utilized in determining the contraction of such bars, when they are moulded in jointed flasks.

The Simultaneous Casting of Duplicate Test-Bars.—Figs. 12, 13 and 14 illustrate the design of flask, mould-board and patterns for moulding and casting double test-bars, with the improved whirl-gate, which the writer has not heretofore illustrated. The method complete is one which the testing-committee of the Western Foundrymen's Association has lately used with the greatest success in obtaining perfect, solid bars, a difficulty which some will remember was experienced with the first designs tested. The plan here shown will enable any ordinary moulder to obtain perfect solid bars, wherever two bars are desired to be cast together at the same moment, and out of the same hand-ladle of iron.

As furnace-men advance in the work of physical tests, many may desire to take up questions which the single cast bar will not permit of investigating, and for this reason the writer has illustrated the system of casting double. Whether the exact plans presented in this paper be adopted or not, the principles upon which they are based cannot be ignored in the attempt to secure true physical tests at the furnace or foundry.

The Effect of Expansion on Shrinkage and Contraction in Iron Castings.

BY THOMAS D. WEST, SHARPSVILLE, PA.

(Contribution to the Discussion of the Physics of Cast-Iron, at the Pittsburgh Meeting, February, 1896.)

THE fact that iron expands when heated, until fusion takes place, and that molten iron is consequently less dense than solid iron of the same grade, is now universally admitted. It was proved by the extensive experiments of Mr. Thomas Wrightson, reported in the *Journal of the Iron and Steel Institute* (1890, No. 1, p. 72, and 1891, No. 1, pp. 75, 109), and, in a manner, is illustrated in heavy founding by the shrinkage of the molten metal, which must be "fed" in order to obtain solid castings.

This decrease in volume requiring "feeding" while the metal is still liquid I call "shrinkage," applying the term "contraction" to the decrease in volume which takes place after solidification, while the iron is cooling to atmospheric temperature. The light-work founder, not having the opportunity to make heavy castings, in which shrinkage can be observed, is apt to confound the two; but they are in fact distinct, and are separated by an act of expansion, which takes place at the moment of solidification. The fact of this expansion was first practically demonstrated by Mr. John R. Whitney, of Philadelphia, Pa., whose experiments are recorded in the *National Car and Locomotive Builder* of May, 1889.

Experiments recently made by the writer indicate that there is a constant relation between this expansion and the preceding shrinkage and forcibly demonstrate the necessity of "feeding" a casting to make its interior solid. This is a matter with which all makers and users of castings have experienced difficulty. The founder being heretofore unable to define correctly the principles involving the urgent necessity of "feeding" has failed to impress the moulder with its importance in making sound castings.

Heavy-work founders and moulders know that hard grades of iron shrink much more than soft grades, a fact for which no satisfactory explanation has heretofore been given. By recent expansion-experiments I have discovered that hard grades of iron expand more at the moment of solidification than soft ones. Fig. 1 is a diagram recording four such experiments.

Fig. 1.

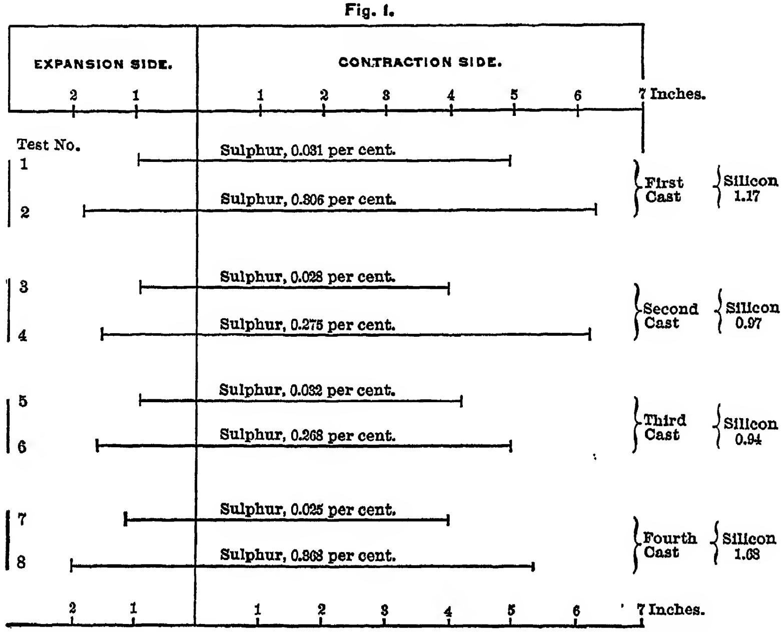


Diagram from automatic records of expansion and contraction, varied by additions of sulphur.

The manner in which the automatic records were obtained will be described further on. It is sufficient to say at present that the scale of inches in the diagram measures the length of travel of the pencils on the long recording-arms of the apparatus employed, not the actual length of expansion. The end of the short arm of each lever, recording actual expansion, travels $\frac{3}{32}$ inch for 1 inch travelled by the pencil, and the length of the test-bars being 48 inches, 1 inch of the expansion- or contraction-record represents an actual expansion or contraction of 3 in 1536, or 0.195 per cent. For the purposes of these experiments, however, the actual expansion or contraction was

not required. The significance of these diagrams is qualitative and comparative; and for this use of them the reading of the pencil-travel in inches is accurate, the apparatus and operation being the same in all the tests recorded. With this explanation I return to Fig. 1.

In each of the four casts shown, two test-bars, 1 by $1\frac{3}{4}$ inches in section and 4 feet long, were cast "open-sand" side by side in the same mould. Tests Nos. 1, 3, 5 and 7 were poured from the respective ladles which brought about 100 pounds of the iron direct from the cupola. These tests comprised the softest iron of each cast and had the least expansion and contraction, as is shown by the diagram. For tests Nos. 2, 4, 6, and 8 the grade of the iron was changed, by means of pouring about half of the hundred pounds contained in the ladle coming direct from the cupola into an empty ladle, the bottom of which was covered with about $\frac{3}{4}$ of a pound of brimstone. The metal in the ladle having the sulphur was then agitated with a half-inch wrought-iron rod until fuming ceased, after which all dross was skimmed from the surface, when each ladle was poured into its respective test-mould. The addition of sulphur hardened the iron in these tests, thereby causing the increased expansion and contraction shown in the diagram.

In Fig. 2, tests Nos. 9 and 10 illustrate another discovery made by this method of comparative tests, namely, that where free expansion is prevented, a greater contraction is effected.

Test-bar No. 9 was cast between iron ends, so arranged that the power of expansion was not sufficient to extend the distance between them, whereas No. 10 had sand ends to compose the mould, which gave full freedom for expansion, the same as in all the other tests displayed in Figs. 1 and 2. The fact that hard grades of iron expand more than soft ones, and the fact that retarding expansion gives rise to a greater contraction than where free expansion is permitted are important as suggesting for works making such specialties as chilled rolls, car-wheels, etc., in which heavy losses are often experienced through chill-checks and cracks, the advisability of adopting expanding and contracting "chills" wherever such may be practicable.

Tests Nos. 11, 12, 13 and 14, in Fig. 2, illustrate the expansion and contraction of different sizes of bars poured in pairs from the same iron. These tests show that large bars expand

so as to increase their interior space more than small ones, thereby calling for the greater "feeding" in massive castings. These tests indicate also that light bars contract more than heavy ones, an element not to be overlooked in proportioning castings so as to avoid internal strains as far as practicable.

The writer's "open-sand" method of casting test-bars affords the means of making comparative tests under varied conditions and gives an excellent opportunity to observe characteristic phenomena at the moment of solidification, etc. In casting test-bars of hard iron, a pronounced shrinkage along the upper surface is often noticed during the period of expansion; and often before expansion is over there may be seen through shrink-holes at the hottest part of the bar (namely, at the point where it was poured) that the interior is still liquid, showing that it is not necessary that the whole body of the casting shall solidify before expansion takes place. In this phenomenon, we perceive also the simultaneous action in the casting of two opposite tendencies; shrinkage going on in some parts, while expansion is occurring in others.

It is the general impression among moulders and founders, that the hotter the iron is poured, the more it will shrink, that is, the more the casting will require to be "fed." This is an error into which the moulder has fallen by reason of the longer time occupied in the cooling or shrinkage of the "hot"-poured metal, and consequently the longer period of "feeding." The total addition of iron required in the "feeding-heads" is no greater with "hot-" than with "dull"-poured iron, unless the "hot"-poured metal has more largely penetrated, fused or strained the walls of the mould.

Numerous experiments have failed to show me any effect produced upon the total expansion by changes in the temperature of the metal when poured. Such an effect would not be naturally expected, since the expansion begins only with solidification, and the temperature of solidification, it is reasonable to say, is always the same for the same grade of iron, under the conditions of these tests; so that, however "hot" iron may have been poured, it will always have a certain temperature when it begins to expand. It is this independence of the extent of expansion as regards the pouring-temperature which constitutes the chief ground of the writer's faith that expansion

may be employed further on, when more is known of it, to determine rapidly the grade of cast-iron or other metals. But it is, of course, clear that expansion will take place sooner in a "dull"-poured bar than in a "hot" one; and again, a light body will expand more quickly than a heavy one, as I have proved by my tests.

The length of the period of expansion varies with the size of the casting. The more massive the casting, the longer the period of expansion. For the bars shown in Figs. 1 and 2, the expansion lasted from one-half to one minute in the smallest bars, and, in the largest bars, from three to five minutes.

The writer would like to learn of expansion-tests made upon steel castings, etc. Since steel exhibits about twice the contraction of cast-iron and likewise great shrinkage, there is reason to expect that a very large expansion would be found in steel, at the time of its solidification.

The relation between the shrinkage and the expansion of solidification may now be indicated. The writer's view is that the apparent shrinkage of liquid metal so familiar to heavy founders is not due chiefly to a change in the specific gravity of the liquid metal as it passes to a solid state, but largely to the effect of the expansion of the solidifying parts of the casting. That is to say, an outer shell of the casting being first formed, its expansion at the moment of solidification necessarily enlarges the interior space to be occupied by liquid metal; and either additional liquid metal must be applied or else cavities and shrink-holes will be found in the interior of medium and heavy castings, by reason of the progressive accretion of the solidifying metal upon the parts already solidified. Such cavities would, on this hypothesis, be likely to be most abundant in the portions which solidified last; and that this is in fact the case, is often proved by practice. Cavities are very liable to occur in the interior of massive castings, and even when castings are properly proportioned, the portion around the "gates," which convey the metal to the mould, is often very likely to be porous or to exhibit shrink-holes, due to the circumstance that the metal solidifies last at these points, and to the attraction of solidifying particles to the already solid mass. This hypothesis explains also the fact that, in heavy castings, poured "hot," shrinkage is not often exhibited in the "feeding-

heads" until long after the pouring, and that when it does commence (which is not before some expansion has taken place, due to parts solidifying), it is often so rapid as to require, for a short period, constant additions of molten metal.

Expansion at the moment of solidification being thus the cause of shrink-holes in castings, the practice (not uncommon among moulders) of placing "risers," not much larger than lead-pencils, so to speak, on massive castings, thinking thereby to make them solid, is to be discouraged as useless. It follows, moreover, that a casting should be "fed" until expansion is ended. It is not while a metal looks "hot" or fluid in a "feeding-head" that attention is specially necessary to secure a solid interior; it is when the metal is thickening or "freezing" in the "feeding-heads" that the greatest attention should be paid to the "feeding." It is a general practice among moulders, at present, to let their "feeding-heads" "bung up," at a time when the greatest effort should be made to keep them open, so as to insure a solid casting. It is at this time, that expansion is taking place, to enlarge the surface-area, and consequently the interior volume of a casting, thereby causing the hottest or most fluid portion of the casting to be robbed of metal, which must be supplied, in order to prevent shrink-holes at all such points.

According to the view here presented, it will be also easy to understand that the resistance offered by the mould may often affect the expansion and shrinkage as well as the subsequent contraction. Whether the power of expansion is as great as that of water in becoming frozen, is, as far as I know, undetermined. I do know that by casting between iron yokes or flask-ends, the longitudinal expansion of the bar may be prevented, as is seen in Test No. 9, Fig. 2. In such a case, of course, it is natural to suppose that the expansion must be in some other direction, and it may increase to a smaller degree the interior space necessary to be supplied with molten metal by feeding. The heat-conducting capacity of the mould, as determining the rate of solidification, may also affect the apparent result. Thus, a casting made in an "iron chill" mould may show less shrinkage than if the same iron had been poured into a sand mould, because, in the latter case, the solidifying iron could have time and opportunity, by reason of the nature of the

mould, to more expand it outward, thus increasing the interior space to be supplied with molten metal as already explained.

To return to the fact discovered by the writer, that hard grades of iron expand in solidifying more than soft grades, it may be said that this is contrary, not only to the general impressions, but also to the current explanation of the fact of expansion, which would ascribe it to the segregation of graphitic carbon. If this were the controlling cause, we should expect

Fig. 2.

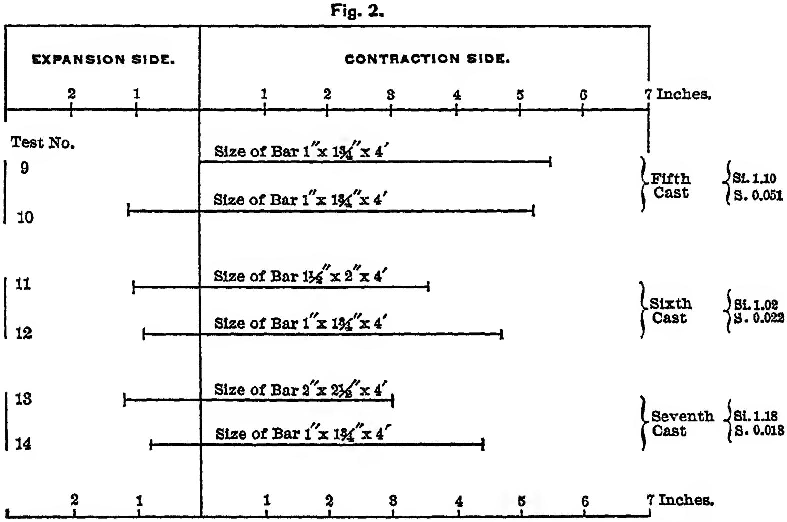


Diagram from automatic records of expansion and contraction, varied by confining expansion, and by using bars of different sizes.

soft irons, which exhibit, after solidification, more graphite, to show the greater expansion.

The formation of graphite is confessedly promoted by silicon, and hindered by the metalloids which "harden" the iron. When these metalloids are present in such proportions as to overpower the effect of the silicon, combined carbon, instead of graphite, is produced in the solidified metal, and the individual grains, crystals, or structural elements of the cast-iron are consequently smaller and more densely packed in hard than in soft grades of such iron. Expansion (and, perhaps, also contraction) would be, therefore, exhibited by a larger number of such structural elements in a given volume of metal. This may explain the greater expansion shown by the hard grades

in Tests Nos. 2, 4, 6, and 8 in Fig. 1, where the largest percentages of the antagonistic constituents, silicon and sulphur, are presented.

But any theory on the subject may be premature. Far more important at this time is the fact itself, which affects so directly our foundry-practice. I attribute the failure to detect it heretofore, to the circumstance that in the every-day work of the founder, the expansion of solidification does not force itself upon his attention. The shrinkage of the liquid mass, requiring "feeding," is obvious enough; and so is the final contraction of the solid mass, for which allowance has to be made in the pattern. But the intervening expansion, not being marked by the final contraction, has been overlooked.

I may here observe that the tests illustrated in Fig. 1 refute the opinion heretofore advanced, that the silicon-contents of an iron can be defined from the final contraction of a casting or test-bar. In all the bars of each cast in Fig. 1 the silicon percentage was practically constant. The variation in contraction, therefore, certainly justifies the assertion that the amount of silicon cannot be thus determined. In fact, the contraction will simply indicate the "grade" of an iron and no more. The metalloids producing this "grade" can only be determined by analysis.

The "grade" of a cast-iron, as I use the term, is a practical name, familiar to heavy founders, though perhaps not capable of precise scientific definition. It is characterized by the degree of hardness, and incidentally by accompanying properties of contraction and of strength. Within the range of ordinary foundry-practice, sulphur and silicon are the two elements most effective in changing the "grade" of cast-iron—the two operating with contrary tendencies, and 1 part of sulphur being equivalent in effect to from 8 to 12 parts of silicon. This statement, ignoring, as it does, the effect of other metalloids, is not presented as comprehensive and exhaustive, but may be useful, nevertheless, as the expression of the practical experience of a founder.

It has been maintained that it is difficult to make cast-iron absorb sulphur and that the founder has no need to fear sulphur in general founding. In the tests shown in Fig. 1, the amount of sulphur in the iron was easily increased by the method described, as is proved by the subsequent analysis. At

all events, I feel sure that up to 0.3 per cent. sulphur can be easily present in cast-iron containing about 1.5 per cent. of silicon, which is a percentage of silicon often permissible and practicable as a maximum in light castings, where the sulphur can be kept below 0.06 in the castings produced. As 0.2 per cent. of sulphur is sufficient to injure or ruin almost any casting made for other purposes than sash-weights, the ability of cast-iron to absorb 0.3 per cent. of sulphur forcibly illustrates the great reason which the founder has to fear sulphur in fuel and high-sulphur iron, and to avoid any method in melting, favorable to the absorption of sulphur by iron in cupola-practice. These considerations are applicable also to the making of iron in the blast-furnace.

The apparatus used for obtaining the expansion- and contraction-records, shown in Figs. 1 and 2, is shown in Figs. 3, 4, 5 and 6. It was designed after much study of the conditions necessary for automatic record of the expansion and contraction of test-bars, and also for the highly important purpose of simultaneous comparative tests.

The figures illustrating this apparatus (which is freely offered, for use, to all who may be interested in the matter) will be readily understood with the aid of the following explanation:

In Figs. 3 and 4 the same letters indicate the same parts, namely:

A, stationary or sliding recording face-plate board; B, float; D, float-receptacle; E, regulator, giving constant head of water; F, supporting arm for the water-supply vessel; H, over-flow pipe; K, L and M, recording-arm levers; N, lead-pencil recorder; O, rubber-band lever-supporter; R, curve-recording face-plate board; S, slide-guides for recording curves; T, revolving sheave-wheel guide and support; U, fulcrum cross-bar; Y, supporter of fulcrum cross-bar.

In Fig. 5 the parts are indicated by letters as follows:

A, counterbalance clock-weight; B, bed-plate, securing the base-board; N, lead-pencil recorder; I, one-day "Pirate" alarm-clock; R, curve-recording face-plate board; S, removable casting-pin; T, casting-pin; U, fulcrum cross-bar; V, clock and recording face-board connecting-shaft; W, casting-pin holder.

In Fig. 6 the parts are indicated by letters as follows:

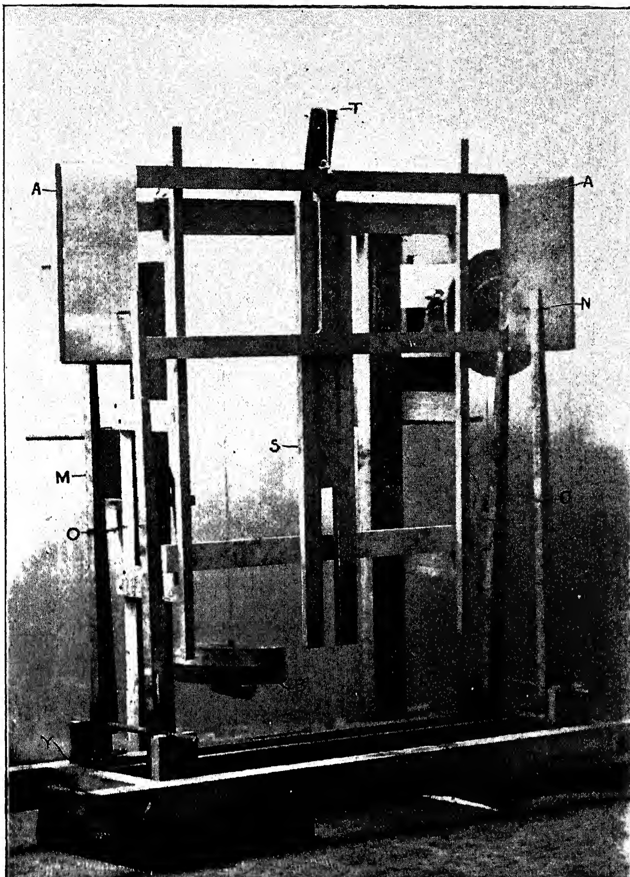
A, expansion- and contraction-end equalizer; B, spring-clasp; D, flow-off recess; E, spring-clasp iron; F, lever-fulcrum bearing; H, casting-pin clasp-opening; K, removable casting-pin.

The levers of this apparatus are so delicately mounted as to be moved by a breath. Those shown in Fig. 6 are of improved form, as compared with those in Fig. 5. As already stated, for every inch travel of the long arm, the short arm, moved by the actual expansion or contraction, travels $\frac{3}{32}$ inch in the straight line. The diagrams, Figs. 1 and 2, were constructed by plotting the sum of the readings given by the pencils at the two ends of the apparatus in straight lines, and consequently give only the total longitudinal expansion and contraction, without indicating rate or alternations. But the apparatus can be employed, with the aid of the float or clock, etc., shown in the figures, to record curves. For a straight line record the face-plate, A, Figs. 3 and 4, is held stationary. To obtain curves, it is gradually lowered at any desired rate by means of the float, B, in the receptacle, D, Fig. 4, a constant head of water being maintained in the reservoir, E, by a supply from a suspended vessel at F, and an overflow-pipe, H. A specially-arranged strong-spring clock might be used instead of the float, B, to lower this face-board uniformly, so as to effect the same end. It is the writer's intention to make a series of experiments giving automatic curve-records, so as to introduce into the results the element of time. Incidentally, such experiments ought to settle the question whether there are, as has been declared, two periods of expansion for cast-iron solidifying and cooling.

The lever-arms, K, L and M, Figs. 3 and 4, are held gently against the face-plate by light rubber bands, secured midway in their lengths at O, so that the very soft pencils at N may record all movements of these arms. The pencil-record may be made on paper, covering the face-plate, as indicated in the figures, or on the bare face of the recording-board.

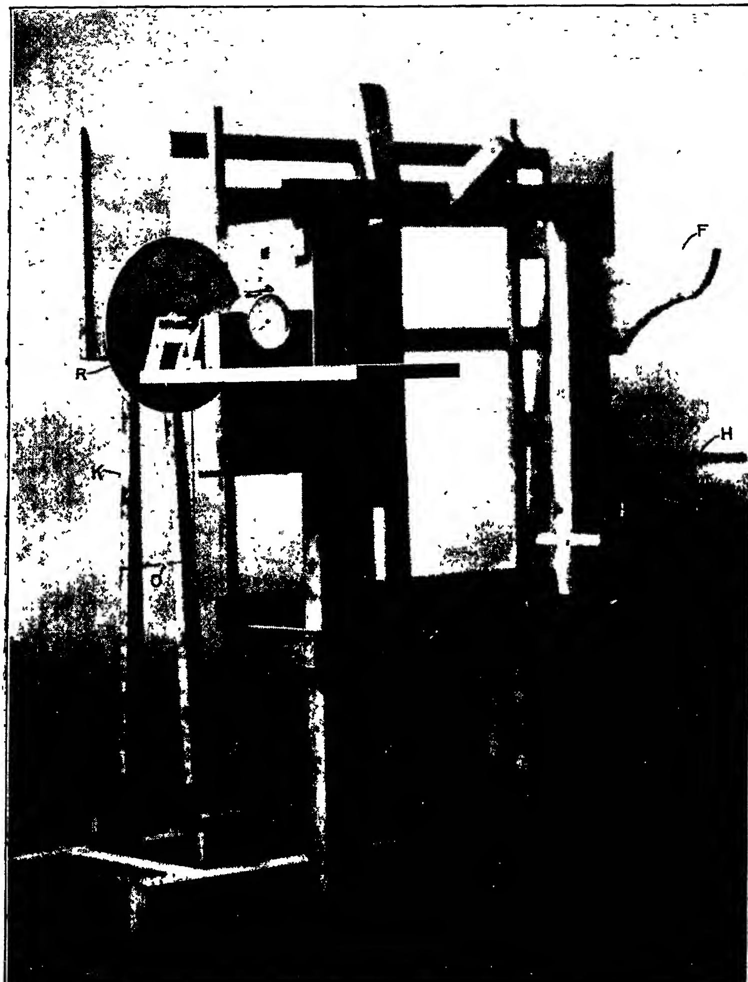
It will be evident that the records of the independent levers at each end of the bar must be added together, in order to determine the total expansion or contraction. Thus, in the case of Test No. 1, Fig. 1, the automatic record of the apparatus would show a travel in expansion of $\frac{1}{2}$ an inch at each end, or 1 inch in all, followed by a contraction of $2\frac{1}{2}$ inches at each

Fig. 3.



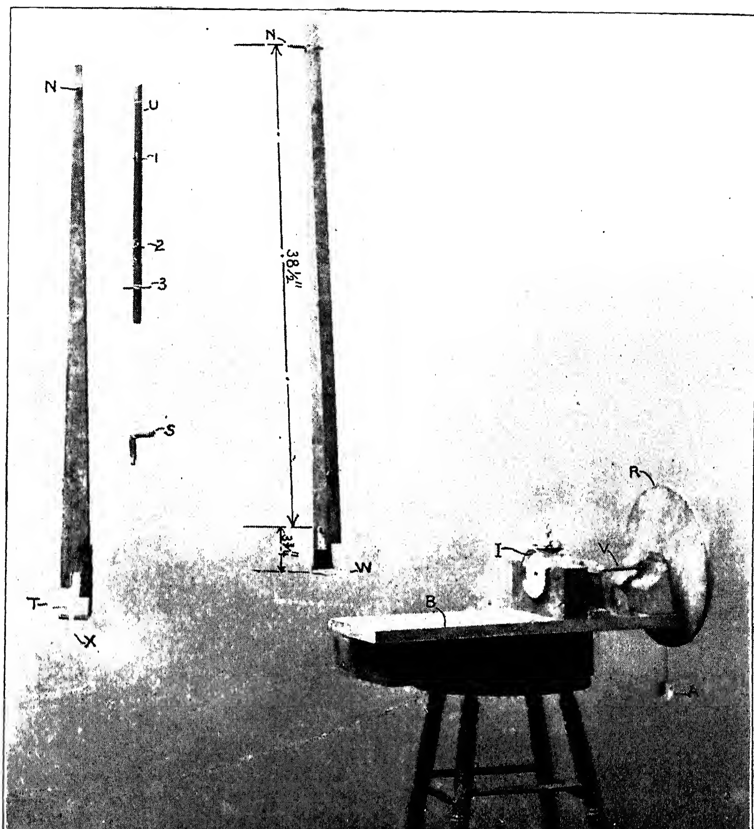
Automatic Recording-Apparatus for expansion and contraction.

FIG. 4.



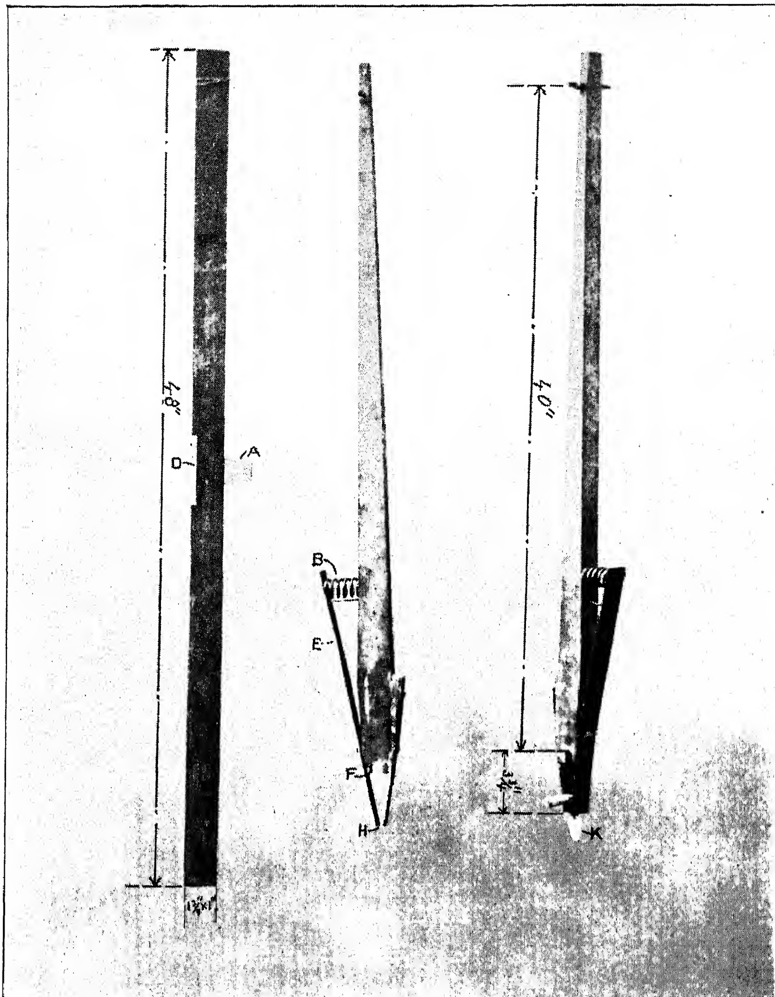
Automatic Recording-Apparatus (seen from opposite side of Fig. 3) with arrangement for record in curves.

Fig. 5.



Details of levers, pin and cross-bar for Recording-Apparatus. Also independent dial for recording expansion- and contraction-curves.

FIG. 6.



Test-bar pattern; and improved form of levers for Recording-Apparatus.

end, or 5 inches in all, not including the retracement of the previous expansion. In other words, after expansion was ended, the bar contracted longitudinally $\frac{18}{32}$ of an inch (each inch of the pencil-line representing $\frac{3}{32}$ inch of the short-arm lever-movement, *i.e.*, of actual extension of the bar); and consequently, the test-bar, 48 inches long as poured, was elongated in solidification to $48\frac{3}{32}$ inches, and then contracted in cooling to $47\frac{17}{32}$ inches, its final length at atmospheric temperature.

The clock shown at I, Fig. 5, with its face-plate, R, can be set independently, with a single recording-lever, to receive on the revolving face expansion- and contraction-curves from one end of the bar only, or it can be supported, as shown in Figs. 3 and 4, so as to record curves in connection with the records made on the stationary or sliding face-board, A.

The whole apparatus is of wood, except the fulcrum-bars, U, Figs, 3, 4 and 5, the casting-pin, S, Fig. 5, and the pin-holding plates, E, Fig. 6. The form of levers shown in Fig. 6 is that which I have adopted since originally photographing the apparatus for the preceding figures. From a study of these levers in Fig. 6 it will be seen that a little pressure on the spring-side at B will instantly release the casting-pin seen at K, thus meeting a difficulty sometimes experienced with the first form of grasp used, which is shown at W, Fig. 5. The $\frac{3}{8}$-inch casting-pins seen at S, Fig. 5, and in position at K, Fig. 6, are made tapering at their points, so that they can be readily moved from a test-bar and used again. They project below the bottom of the spring-clasp to about the distance indicated in Fig. 6, so as not to touch the bottom of the mould when being cast in the bar for the purpose of causing the levers to record sensitively any movements due to expansion or contraction after the bars are poured.

At the left of Fig. 6 is seen the form of pattern used for moulding the test-bars. The projection at A is cast on, as shown, so as to insure equal action in recording the expansion and contraction at each end of the bar. At D is seen a recess, which gives a guide to make the same in the mould, so that in pouring the bars "open-sand," the metal will "flow off" at this point when it comes to that level, and thereby insure all bars being cast closely to the same thickness.

The Mobility of Molecules of Cast-Iron.

(A Contribution to the Discussion of the Physics of Cast-Iron.)

BY A. E. OUTERBRIDGE, JR., PHILADELPHIA, PA.

(Pittsburgh Meeting, February, 1896.)

It has been generally accepted as a fact that cast-iron, under the influence of repeated shocks, becomes brittle, and will finally break under a blow which otherwise it would have withstood. It will probably surprise metallurgists, therefore, to learn that experiment disproves the supposed fact, and establishes exactly its opposite.

The result of about a thousand tests of bars of cast-iron of all grades, from the softest foundry-mixtures to the strongest car-wheel metal, enables me to assert with confidence that, within limits, cast-iron is materially strengthened by subjection to repeated shocks or blows.

It is very well known that the usual process of annealing castings—such, for example, as car-wheels—increases their strength by relieving cooling-strains. But it is not well known (if known at all prior to this announcement) that the molecules of cast-iron are capable of movement—for they do not touch each other*—without the necessity of heating the casting, and that they can thus rearrange themselves in comfortable relation to their neighbors, and relieve the overcrowding near the surface of the casting. In more technical words, a molecular annealing may be accomplished at ordinary temperatures which will release the strains in the castings, precisely as does annealing by slow cooling in heated pits or ovens. A statement so surprising should not be made without sufficient data to establish its correctness beyond cavil, since it is contrary to former

* "In order to comprehend the modern idea of the nature of matter, we should try to realize that the molecules composing even the most dense solid substances with which we are familiar—such as gold, platinum, etc. —are not in contact, and are free to move within certain well-defined limits."—Lecture on "Matter," by the writer, *Journ. Franklin Institute*, September, 1885, vol. xc., p. 184.

belief, certain to be questioned, and properly so. Before proceeding to give a record of the experiments which have been made, and which can be readily repeated by any one, a brief history of the origin of the first observation leading thereto may be interesting.

In 1883 (being at that time engaged in metallurgical work at a large car-wheel establishment), I noticed that chilled cast-iron car-wheels rarely cracked in ordinary service after having been used for any considerable time; if wheels did not crack when comparatively new, they usually lasted until worn out or condemned for other causes. No application was made of this observation at that time, further than to institute a careful investigation of the condition of the annealing-ovens when some new wheels were returned cracked, under the supposition that the wheels were not well annealed, and an equally careful revision of the iron-mixture to ascertain whether the fault lay therein.

In 1894, a large number of "transverse test-bars," 1 inch square and 15 inches long, accumulated in the foundry of Wm. Sellers & Co., Incorporated; and, to expedite the cleaning of sand from their surfaces, they were all thrown into an ordinary "tumbling-barrel" with other castings, and knocked about for several hours. When these test-bars were broken upon the transverse testing-machine, and the records were tabulated, I noticed with surprise that the average strength of the entire series was considerably higher than was usual with similar iron-mixtures. This difference was fortunately so marked as to cause a careful inquiry—first, into the condition of the testing-machine; then, as to the chemical composition of the metal in the bars. The machine was found to be in good order, and the metal was normal. A card pattern was then made, upon which twelve test-bars could be moulded side by side in one flask, and poured from one runner. Six of these bars were placed in the tumbling-barrel, the other six were cleaned of adhering sand with an ordinary wire brush, and the twelve bars were broken upon the machine. *All of the bars which had been subjected for about four hours to incessant blows in the tumbling-barrel were stronger than their companion bars*—the actual gain varying from 10 to 15 per cent. This metal was soft foundry-iron.

These tests were repeated on several consecutive days with similar results, while various theories were suggested, and

clues were followed, to detect the hidden cause of this strange fact.

One plausible explanation offered was, that the rubbing of the bars together in the tumbling-machine slightly rounded the corners, and thus prevented a starting-point for a "check" or break of the bar under the strain in the testing-machine.

This theory was soon overthrown by tests. The corners of six bars were rounded by filing (the companion-bars not being filed); all of the bars were then cleaned with a wire brush and broken upon the transverse testing-machine, and there was no apparent gain in strength in the bars with rounded edges. Round test-bars (1⅛ inches in diameter, 15 inches long) were then poured from one ladle of iron. Some of these were cleaned in the tumbling-barrel, and all that were so treated proved to be much stronger than the companion-bars which had been merely cleaned with a wire brush.

This process of eliminating false theories was continued, until finally a new explanation occurred to me, and simultaneously a convincing test of its accuracy suggested itself. The explanation, as indicated in the title of this paper, is the mobility of the molecules of cast-iron, at ordinary temperature, when subjected to repeated shocks.

The crucial test referred to consisted in subjecting 6 bars to 3000 taps each with a hand-hammer *upon one end only of the bar.* All the bars so treated showed a gain in strength equivalent to the gain exhibited by bars which had been subjected to blows over the entire surface for several hours in the tumbling-barrel. Here was a new revelation, of scientific interest to the metallurgist, and suggesting to the founder the possibility of annealing castings at ordinary temperatures by availing himself of this molecular mobility. It proves also that we have for many years been unconsciously accomplishing this beneficial result, at least partially and irregularly, by tumbling small castings in a revolving barrel, merely for the purpose of conveniently cleaning them from adhering sand.

Another interesting fact was incidentally brought out in this investigation, namely, that the strain caused by cooling, and the consequent weakening, exists even in the smallest castings, where we would naturally expect that the cooling would be practically uniform throughout the section; moreover, that

even in bars of uniform section, such as 1-inch test-bars, the weakening by cooling-strains sometimes amounts to more than 10 per cent. of the ultimate stress. Repeated tests show that ½-inch bars, in which the fracture is comparatively uniform to the eye throughout the section, are subject to the same law. Other tests show that the comparative difference in transverse strength between bars which have been hammered on their ends, or otherwise subjected to the process of molecular annealing by vibration when cold, and companion-bars cast in the same flask, not so improved, depends to a certain extent upon the number and force of the blows, and to a still greater degree upon the grade of cast-iron tested. For example:

1. Greater relative difference is found in hard mixtures, or strong iron, than in soft mixtures, or weak iron.

2. Greater relative difference is found in 1-inch bars than in ½-inch bars, and somewhat greater difference in 2-inch bars than in 1-inch bars.

Impact-Tests.—The foregoing experiments having been repeated sufficiently often to satisfy me of their absolute reliability (and really remarkable uniformity), a new series of tests was commenced for the purpose of ascertaining how many blows were required, and approximately what force was needed to accomplish the desired object of relieving cooling-strains. A new machine was constructed for this purpose, and light was soon thrown upon these questions, while still other important questions were suggested and answered by the use of the same machine.

The impact-machine first used was an old one, and consisted of a weight fastened to an arm swinging in a graduated arc. The friction of the pivot and the crude construction of the machine precluded even an approximately accurate measurement of the force of the blow delivered.

The new machine consisted of a frame or yoke, marked in inches, and a wedge-shaped weight, adapted to the size of the bars to be tested, which was raised vertically to any desired number of inches, and, when released, fell by gravity in free space, striking the bars in the center between the supports, which were 12 inches apart. A 14-pound weight was adopted for testing 1-inch bars.

Impact testing-machines have long been used, and car-wheels

are accepted or condemned according to tests made upon such machines. It has always been maintained that each blow of the "drop" weakened the casting, and that the final blow was only a record of the residual cohesion remaining in the metal, after previous blows had proportionately weakened it. In the case of a thoroughly annealed car-wheel this reasoning may be sufficiently correct; but as applied to impact testing-machines used for testing unannealed bars, the argument is absolutely fallacious. In such cases, *the impact-machine is itself a means of molecularly annealing test-bars*, as I will now demonstrate.

Impact-Experiments.—Six of the 1-inch square test-bars, cleaned with the wire brush, were broken upon the impact-machine by dropping the weight from a sufficient height to break each bar at the first blow. The six companion-bars, also cleaned with the brush, were then in turn subjected to blows, numbering from 10 to 50 each, of the same drop-weight, falling one-half the former distance, these blows being insufficient to break the bars. The weight was then permitted to fall upon each of these bars in turn from the height at which the six bars previously tested had been broken at the first blow. *Not one bar broke.* Two, three, six, ten, and in one case fifteen, blows of the same drop from the same extreme height were required to break these bars. In another similar case the weight was dropped once from the former height, then raised by inches until four more blows, each being one inch higher than the last, had been delivered before breaking the piece. Subsequent tests showed still greater gain in strength.

The next experiment with the impact-machine was designed to test molecularly-annealed bars from the tumbling-barrel, in comparison with untreated companion-bars, under one heavy blow. It was found that a blow of sufficient force to break the unannealed bars with one fall of the weight must be repeated from five to twenty times (depending mainly upon the nature of the iron-mixture) to break the molecularly-annealed companion-bars. By careful experiment in the manner described, it was shown that the ultimate strength of the bars which had not been through the tumbling-barrel could be increased, by successive slight blows upon the impact-machine, to an equality with their companions.

The experiments here related in a running conversational

narrative, form a part of the daily records of metallurgical work at the foundry of William Sellers & Company, Incorporated, Philadelphia; and the aggregate number tabulated is very large. All the tests corroborate fully the statements here made; and they are susceptible of repetition and confirmation by any one interested in work of this character. They form a part of a long series of investigations (extending over a period of fifteen years) upon the relation between the physical nature and chemical composition of cast-iron; they have served to throw light upon phenomena hitherto obscure, connected with the design, construction, etc., of castings; and, as it is believed that they contain the germ of a new scientific discovery valuable in its principle to all workers in these fields, these brief notes are presented to the Institute in response to the invitation of its Secretary, and through the courtesy of the firm of William Sellers & Company, Incorporated, for whose benefit and at whose cost they were primarily made.

In conclusion, it should, perhaps, be observed, to avoid the possibility of misunderstanding, that the molecular annealing of cold cast-iron by successive slight shocks differs from annealing by heat, in that it has no power to change the condition of carbon in the casting or to alter the chemical constitution in any way. All that is claimed is, that every iron casting, when first made, is under a condition of strain due to difference in the rate of the cooling of the metal near the surface, as compared with that nearer the center, and also to difference of section; and further, that it is practicable to relieve these strains by repeatedly tapping the casting, thus permitting the individual metallic particles to rearrange themselves, and assume a new condition of molecular equilibrium.

The large number of tests made and the remarkable uniformity of the results obtained warrant me in making these statements with full confidence that the repetition of such experiments, even under less favorable conditions for accurate observation than I have enjoyed, will convince others of the correctness of my conclusions, and will, perhaps, establish a new law of the physics of cast-iron.

A few practical deductions of universal application may be drawn from these observations.

1. Castings such as hammer-frames, housings for rolls, cast-

iron mortars or guns, which are to be subjected to severe blows or strains in actual use, should never be suddenly tested to anything approaching the severity of intended service.

Quantitative tests made upon the impact-machine prove that the molecules of cast-iron rearrange themselves under reasonably few shocks, so that it is perfectly practicable to molecularly anneal such castings when cold. Pulleys, and indeed all castings, are subjected in every-day service to this process of molecular annealing; and old castings are therefore more reliable than new ones, unless they have been misused.

It is not impossible that the same law applies to steel-castings and perhaps to all metal-castings, and that in testing new guns, each preliminary small charge of explosive material, subjecting the casting to comparatively moderate shocks, enables the gun to relieve itself of internal strains, and eventually to withstand with safety shocks which would have destroyed it without this precautionary measure. This, however, is mere theory, and must not carry the weight of the arguments regarding cast-iron, which are clinched by a thousand actual tests.

2. Strong iron castings, and castings of irregular section, have greater initial strains than soft iron castings or castings of comparatively uniform section; and it is, therefore, more important to subject the former to gradually increasing shocks until the strains are relieved by the movement and rearrangement of the molecules.

Tables.—The tables here given show the results of tests for transverse strength of test-bars, of different section and widely different grades of iron, which have been subjected to this process of molecular annealing, and also tests made upon the impact-machine with test-bars cast in the same moulds. To avoid the unnecessary duplication of figures, a few individual tests only are here tabulated, but they represent the average of probably a thousand records.

The largest apparent gain shown in Table I. in the transverse strength of companion-bars of 1-inch section, is 525 pounds (see last line of the table), or very nearly 19 per cent.

It is evident, even without plotting all the tests, that they would show a gradually ascending curve having direct relation, first, to the character of the alloy of iron; and second, to the

number of blows given, up to the point when strain is relieved—beyond which an increased number of blows does not increase the strength of the bar. Furthermore, it may be noted that these tests with the impact-machine indicate the existence of a similar law with relation to the ability of the metal to resist sudden and severe shocks.

Similar observations apply to the tests recorded of half-inch and 2-inch bars. All the 2-inch bars were broken upon the testing-machine of Messrs. A. Whitney & Sons, by their operator, who was at first ignorant of the object of the tests. Subsequently, similar tests were made by that firm with their car-wheel iron, and identical results were obtained by them.

The tabulated tests have been selected from the daily records of experiments made at the machine-tool works of William Sellers & Co., Incorporated, Philadelphia.

In Table I., the left-hand columns show the breaking-strain and deflection of test-bars of different kinds of cast-iron, cleaned in the ordinary manner with a wire-brush. The right-hand columns show the tests of companion-bars, cast from the same runner, molecularly annealed by being subjected to shocks in the tumbling-barrel, or by tapping on one end with a light hammer, as noted under "Remarks;" also, a few tests made with round bars $1\frac{1}{8}$ inches in diameter, cast on end.

Table II. gives a few of the tests with 1-inch bars, made upon the new impact testing-machine, using a 14-pound weight. The tests marked A represent bars cleaned with a brush, and those marked B, companion-bars, molecularly annealed in the manner described.

Table III. gives a few records of 2-inch test-bars, broken by A. Whitney & Sons upon their machine. The tests marked A were cleaned with the brush; those marked B were molecularly annealed in the tumbling-barrel.

Other experiments have suggested themselves in the course of this investigation; and my paper, therefore, is not presented as an exhaustive, but as a tentative treatment of this interesting and, I believe, novel line of research, which is not incapable, even in its present stage, of some practical application.

Table I.

Unannealed. Breaking Strain. Pounds.	Unannealed. Deflection. Inches.	Molecularly Annealed. Breaking Strain. Pounds.	Molecularly Annealed. Deflection. Inches.	Remarks.
2350	.13	2850	.14	Close grain.
2025	.12	2300	.14	Open “
2125	.13	2275	.14	“ “
2275	.13	2400	.14	“ “
2525	.14	2850	.15	Close “
2175	.13	2500	.15	Open “
2100	.13	2375	.14	“ “
2025	.12	2300	.14	“ “
2775	.14	2900	.15	Close “
2150	.13	2250	.13	Hammered on end of bar.
2550	.14	2925	.15	“ “ “
3000	...	3200	...	Round bars 1⅛ inches diam.
3000	...	3150	...	Hammered on end of bar.
2150	...	2450	...	Round bars 1⅛ inches diam.
2100	.13	2425	.15	Open grain.
2100	...	2400	...	Round bars 1⅛ inches diam.
2050	.12	2400	.14	Open grain.
2875	.14	3100	.15	Close “
2175	.13	2500	.15	Open “
2150	.13	2350	.13	“ “
2675	.13	3000	.14	Close “
2775	.14	3300	.15	“ “

Table II.

A. 1 blow of 14-pound weight falling 13 inches broke the bar. Open grain.
B. 8 blows “ “ “ 13 “ “ “
A. 2 “ “ “ “ 14 “ “ “
B. 9 “ “ “ “ 14 “ “ “
A. 2 “ “ “ “ 14 “ “ “
B. 14 “ “ “ “ 14 “ “ “
A. 2 “ “ “ “ 15 “ “ “
B. 15 “ “ “ “ 15 “ “ “
A. 3 “ “ “ “ 15 “ “ “
B. 8 “ “ “ “ 15 “ “ “
A. 2 “ “ “ “ 15 “ “ “
B. 7 “ “ “ “ 15 “ “ “
A. 3 “ “ “ “ 13 “ “ Close grain.
B. 4 “ “ “ “ 15 “ “ “
A. 2 “ “ “ “ 13 “ “ “
B. 10 “ at 13 inches, 5 at 15 inches, 5 at 17 inches, 3 at 18 inches, broke the bar. Close grain.
A. 1 “ of 14-pound weight falling 14 inches broke the bar. Open grain.
B. 20 “ at 13 inches and 3 blows at 14 inches broke the bar. “
A. 3 “ of 14-pound weight falling 13 inches broke the bar. Close grain.
B. 15 “ “ “ “ 14 “ “ “
A. 8 “ “ “ “ 12 “ “ “
B. 50 “ at 12 inches and 10 blows at 15 inches failed to break the bar.

Note.—The last bar, after withstanding these blows, was broken upon the transverse testing machine at a strain of 2975 pounds.

TABLE III.

	Breaking Strain in Pounds.	Grade of Iron.
A,	20,400	Close grain.
B,	21,600	"
A,	13,600	Open grain.
B,	15,800	"
A,	14,000	"
B,	14,800	"

I have suggested, and adopted throughout this paper, the hypothesis of the mobility of the molecules of cast-iron, resulting in an effect similar or analogous to the effect of annealing by heat, because it seems to be warranted by the facts developed by the experiments described; it is of course possible, in our *a priori* ignorance of the laws governing atoms and molecules, that the theory may not be correct, but the tentative propounding of a probable hypothesis, by inciting to a more extended course of experiment, along different related lines of investigation, often leads to establishing or disproving the theory, and thus adds to our stock of positive knowledge.

Note on Carbon-Bricks in the Blast-Furnace.

BY R. W. RAYMOND, NEW YORK CITY.

(Pittsburgh Meeting, February, 1896.)

IN connection with the discussion of the paper of Mr. James Gayley, presented at the Baltimore meeting, in February, 1892, on "The Preservation of the Hearth and Bosh-walls of the Blast-furnace,"* I offered a translation of a letter received from our distinguished honorary member, Professor P. Ritter von Tunner, concerning the use of carbon-bricks at the Donawitz furnace, then (February, 1892) in the third month of its campaign. The purpose of this brief paper is to put upon record later results of practice with carbon-bricks, both in this country and abroad.

* *Trans.*, xxi., 102.

Before doing this, however, I wish to correct an error in my translation of Prof. Tunner's communication, to which my attention has been called in the course of later correspondence. Namely, I was misled by some incidental phrases employed by him, to conclude that the whole of the Donawitz furnace, above as well as below the boshes, was lined with carbon-bricks. Later information has shown me that this was not the case, and that the "upper" and "lower" linings of carbon-brick, then understood by me as respectively above and below the bosh-line, were intended to mean the lining of the hearth or crucible, above and below the tuyeres. Prof. Tunner's reference to the abrasive action of descending material* must therefore be understood as referring to the walls of the smelting-space above the tuyeres. I make this correction because I think it should never be deemed too late to confess and rectify errors in our *Transactions*. So far as I am now aware, this particular error has not had, and may not have, any important effect. Yet it is well, by correcting it, to put on record the fact (which I have ascertained also by direct correspondence with experts abroad) that carbon-bricks have not been employed hitherto, in Germany or Austria, for the lining of the whole of a blast-furnace. The question, therefore, whether they could be so manufactured as to be superior to chamotte-bricks (*i.e.*, ordinary fire-bricks) or other materials, for the lining above the boshes, is one concerning which no light is offered by foreign experience.

With this explanation, I proceed to give the latest information (November, 1895) concerning the Donawitz furnace, for which I am indebted to the management, through the courteous intervention of our honorary member, Prof. Hanns Hoefer, of Leoben, Styria.

According to this statement, the carbon-bricks lining the hearth were laid within a strong iron mantle. The thickness of the hearth-wall was 1 meter, in two interlocking layers of 600 and 400 millimeters respectively. The mortar consisted of 3 parts coke and 1 part clay.

Before blowing-in, the carbon-bricks were protected against burning during the heating-up, with a 150-mm. layer of cham-

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* *Trans.*, xxi., 121.

otte-bricks. The bottom of the hearth was made of chamotte-bricks.

The furnace has now run four years, with an average annual product of 55,000 tons. The carbon-bricks are still tolerably long (over 700 mm.), and justify the expectation of a considerable campaign yet to come.

Mr. James Gayley advises me that there is nothing new to be added on the subject from experience at the Edgar Thompson furnaces. There is at these works one furnace with carbon-bricks in the bosh-wall, which was put in blast in 1893, and is still running, and giving satisfactory results. The commercial value of the carbon-bricks, however, cannot be determined until the blast is ended, when it will be possible to compare the advantages resulting from their use with the extra cost involved. Pending such determination, the company is not using these bricks in other furnaces.

The Assay by Prospectors of Auriferous Ores and Gravels by Means of Amalgamation and the Blowpipe.

BY WILLIAM HAMILTON MERRITT, ASSOC. R.S.M., ETC., TORONTO, CANADA.

(Pittsburgh Meeting, February, 1896.)

AT the Atlanta Meeting in October, 1895, a short paper was presented by Mr. R. W. Leonard on the "Assay of Auriferous Ores and Gravels by Amalgamation and the Blow-pipe" (*Trans.*, xxv., 645), embodying the results of some very conclusive tests as to the value of the process. His arguments regarding the desirability of the process appear to be indisputable, but the practicability of the field-outfit used by him is open to question.

The Province of Ontario, in which Kingston is situated, is still largely unexplored; and at the Kingston School of Mining we give much attention to the instruction of prospectors and the testing of ores.

To this end, not only is there a small mill (to which Mr. Leonard alludes) attached to the school, but prospectors' classes

are held both at the school and at centers in the mineral-bearing districts. In these classes the testing of auriferous ores by pan-amalgamation, assay and blowpipe have been carried on ever since the classes were handed over to the School of Mining by the government.

The method which I have adopted differs in some particulars from that pursued by Mr. Leonard. I shall point out some of the differences, not that I claim any originality in such an old established practice, but to show that the field-testing of gold-ores (which is very properly characterized by Mr. Leonard as important) has not been overlooked in the School of Mining, and also, that a very much cheaper outfit than that described by him can be used with practical satisfaction for the same purpose.

In the first place, after sampling the ore, we treat two pounds, the one-thousandth part of a ton being a convenient quantity to operate upon. We pass the ore through a 40- or a 60-mesh sieve, generally the latter; but if it be known that the ore is to be treated in a mill using a screen with the former mesh, then the use of that size will give a fairer test.

We use an ounce of mercury, and I find (as stated by Louis) that the results are better when sodium amalgam is used, as the mercury does not flour so much, and also that a little sodium helps immensely to gather the very fine globules of quicksilver toward the end of the panning. After panning, the mercury is weighed to see if it all has been recovered. If there is a loss, the pulp is repanned, or a pro-rata loss of gold is allowed for.

Instead of grinding the pulp after the mercury has been mixed with it, I use a pestle made of soft wood, for I consider that thorough mixing (say for an hour) will give as intimate a contact as is obtained in the stamp-mill, and that (as the test is really to determine what we may expect from the mill) this result is no more than we wish for. Fine grinding with the mercury would probably give a higher result. This it may be desirable to ascertain by subsequent investigation; but in the first instance we seek the result we may expect from the stamp-mill.

After panning and getting first the mercury and afterwards the concentrates, and estimating the percentage of concen-

trates in the ton of ore, the system we use differs considerably from that described by Mr. Leonard. The greatest difficulty in prescribing any quantitative test for the prospector lies in his inability to purchase, or transport, an expensive balance weighing to 0.1 milligramme, as required by Mr. Leonard's method. We therefore follow a ruder method, which is, nevertheless, practically useful. In operating upon 2 pounds of pulp, every grain of gold or bullion obtained will represent approximately 2 ounces to the ton of ore (more precisely, 2.08 ounces Troy of gold or bullion in 2000 pounds avoirdupois of ore). An excellent little balance with sliding weight, capable of weighing from 0.1 up to 5 grains, can be bought for $3. As every grain of gold obtained represents practically 2 ounces per ton, it will be seen that a gold-ore can be determined running as low in free gold as $3.60 per ton (the bullion being estimated at $18 an ounce) by means of this $3 balance. Indeed, it is quite possible to adjust the weight half-way between two divisions and get a result below $2, as we have done in our testing.

Again, in the matter of a retort for retorting the small quantity of mercury, we find that a little sheet of Russia iron can be worked by a tinsmith, for a few cents, into a deep cup-shaped retort, with a smooth rounded bottom and corrugated sides. If necessary, this can be unbent and flattened, should any difficulty be found in getting out the gold, which seldom occurs.

To avoid the fumes, or to save the mercury, a hollowed-out potato answers the purpose of a cover and condenser. The sponge of bullion is preferably taken out of the retort, mixed with lead and fused by blowpipe on charcoal or in a capsule, flux being added for refining after the first fusion. Cupellation is done in a clay pipe. A good idea of the bullion will be obtained by the color of the button, but to be exact it can be parted in the usual manner and both gold and silver estimated.

The amalgamation can be done in a miner's pan (kept for that purpose); therefore the whole outfit, balance included, only costs a few dollars, and is within the limit of financial possibility of all prospectors. The cheapness and portability of the outfit is also not objectionable to the mining engineer in the field.

We generally content ourselves with testing the concentrates

for gold and silver by the blowpipe, I may say, qualitatively, without measuring the bead, going on the principle that any gold bead which is visible to the naked eye, or a fair-sized silver bead, are evidences that the concentrates are worth a more careful assay. However, for the more advanced students, we do quantitative work by measurements with the Plattner's ivory button-scale, following the plan recommended by Mr. E. L. Fletcher in his excellent little work, published by Wiley & Co.

In conclusion, I may observe that gold-ores which we have tested in our prospectors' classes have yielded results from $115 free gold per ton, with $689 concentrates (or $8 in concentrates per ton of ore), down to $3 free gold per ton and nothing in concentrates. One ore was determined down to $1.50 per ton, free gold. Some samples have proved to contain nothing, but we naturally avoid using them for class-instruction. A sample of alluvial sand from this province, treated in the manner above described, was proved to carry $2 per ton in free gold. As might be expected, its concentrates carried nothing.

In the above cases, the concentrates were regularly assayed. As an example, however, of results obtained in a prospectors' class, when the value of the concentrates was determined by the Plattner scale, the following instance of a rich free-milling ore is given.

Two pounds of the ore yielded 4.3 grains of bullion, which was not parted, but, being very yellow, was estimated at $19 per ounce, making for 1 ton of ore 8.6 ounces, valued at $163.40.

The amount of concentrates obtained from 2 pounds of ore was 0.76 ounce, or $\frac{1}{42}$ of the weight of the ore or pulp. That is to say, from 42 tons of ore 1 ton of concentrates would be obtained.

The blowpipe-assay of the concentrates (3 grains being taken) gave, as measured on the Plattner scale, 2 ounces of bullion per ton, or $38, which, being divided by 42, gave 90 cents as the value of gold in concentrates per ton of crude ore. The result is therefore, per ton of ore:

Available by free-milling, ...	$163.40
In concentrates, ...	0.90
Tailings, ...	not determined.

Cost and Detailed Description of Apparatus.

The *panning* outfit catalogued below, including sufficient supplies of reagents, etc., for any ordinary prospecting-trip, will cost about $7.50. It may consist of:

1. Glass-stoppered bottle, containing strong nitric acid. (This can be carried in a "patent lightest-weight liquid-mailing case.")
2. Two gold-pans; one to be used for mercury only.
3. Mercury, about 1 pound.
4. "Travellers' letter- and parcel-balance" hand-scale, weighing 0.25 to 12 ounces, for weighing mercury and pulp; cost, 30 cents.
5. Balance, hand-scale with sliding weight, very sensitive, from 0.1 to 5 grains; cost, $3.
6. Small Russia sheet-iron retort, and sheet of Russia iron 1 foot square (with hole for retort in the center), for supporting the retort.
7. Small porcelain dish or thimble.
8. Iron mortar and pestle; cost, 90 cents.
9. Brass wire 60-mesh sieve; cost, 50 cents.
10. A little sodium carried in naphtha, in a wide-mouthed bottle, in a "patent lightest-weight liquid-mailing case."
11. Wooden pestle.
12. Sheet- or shot-lead (pure, if possible).
13. Borax.
14. Soda.
15. Blowpipe, cost, 25 cents.
16. Bone-ash.
17. Clay pipe for cupelling.
18. Charcoal.
19. Candles.

For *quantitative* determination of value of concentrates by measurement with Plattner's ivory scale (cost $3), a sufficient outfit, including the scale, can be obtained for $5, if the prospector makes his own little anvil, pestle and guard and pincers, and gets a small cheap hammer. He will need in addition (included in the $5) only a Fletcher blowpipe furnace, clay crucibles and capsules, a spirit-lamp and some litharge.

For *qualitative* work a prospectors' simple blowpipe outfit might comprise:

1. Knife.
2. Magnifying-glass.
3. Blowpipe.
4. Charcoal.
5. Candle.
6. Old scissors.
7. Pincers.
8. Steel anvil, $\frac{1}{2}$ by $1\frac{1}{2}$ by 2 inches.
9. Pestle and guard.
10. Small hammer.
11. Magnet.
12. Borax.
13. Soda.
14. Litharge.
15. Bone-ash.
16. Clay pipe for cupel.
17. Round-headed bolt for making cupels.

To which may be added platinum-wire, spirit-lamp, micro-cosmic salt, cobalt nitrate, three-cornered file and glass tubing. The total cost need not greatly exceed $1.

Therefore for the entire panning, qualitative and quantitative field-outfit for purposes above indicated the cost need not exceed $14, and with it the prospector, or indeed the mining engineer, can with practice obtain in most cases valuable information in the field concerning the ores of the precious metals.

Weight of Apparatus.—The weight of complete outfit, including the panning, qualitative and quantitative outfits (avoiding duplications in above lists), may be about:

	Pounds.	Ounces.
Two pans, ...	3	12
Mortar and pestle, ...	11	
Remaining articles, including mercury and other ingredients, ...	5	4
Total weight, ...	20 pounds.	

Vein-Walls.

BY T. A. RICKARD, DENVER, COLORADO.

(Pittsburgh Meeting, February, 1896.)

FROM time immemorial the fissure-vein has been held the simplest type of ore-deposit. The prominence given to it by Cotta and his disciples, from their study of the mines of the *Erzgebirge*, is impressed upon technical literature; and, in consequence, the ores which carry the valuable metals have been supposed to occur mainly in fissures, cleaving the rocks in diverse directions, and the noblest type of vein has been deemed that which cut across the country independent of its structure, whether evidenced as bedding, foliation or cleavage, and which was identified with rents produced in the rocky crust of the earth.

As so conceived, the vein was a fissure filled with ore, extending through the country for a varying distance, and continued downward to a depth more or less proportionate to its longitudinal extent. The vein-material was bounded by an encasement of rock, and those immediate surfaces which limited it on either side were called "walls."

These primary conceptions have become modified by the experience of modern mining in widely separated regions. The study of lode-formations has led to the recognition of notable departures from the supposed normal structure of the veins of Saxony and Cornwall, the two classic homes of early economic geology.

Typically the walls of a vein are conceived as parallel rock-planes enclosing the ore; the upper one being called the "hanging-," and the lower the "foot-wall."*

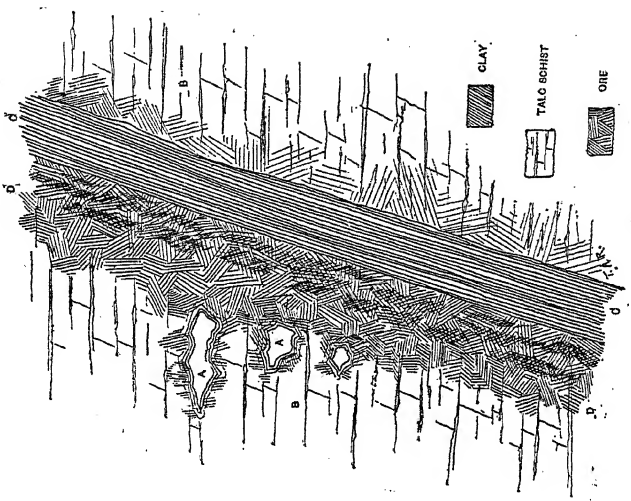
Walls are rarely alike. Even where a vein traverses a homogeneous formation, such as a massive crystalline rock, it is usually found that the surface which bounds it underneath

* The French equivalents are *le toit*, "the roof," and *le mur*, literally, "the wall." In German, *das Hangende* and *das Liegende*.

differs from that which limits it overhead. This is to be ascribed to the effect of the agencies which brought about the deposition of the ore. The action of underground waters tends at first to affect both equally; but in many cases probably the solutions, as they slowly ascend along the line of fissuring, are prevented from penetrating into the encasing rock by the occurrence of an impermeable covering of clay, due to abrasion, which may line either wall, but, because of gravity, generally accompanies the under one. Similarly we are justified in supposing that the deposition of a mineral deposit may form a coating which would serve to protect the foot-wall from the corroding effects of chemical action. The activity of the mineral-bearing current thus becomes diverted in its greatest intensity toward the upper wall, where the decomposition of the rock-surface may be followed by its disintegration so as to cause the exposure of fresh faces for further dissolution.

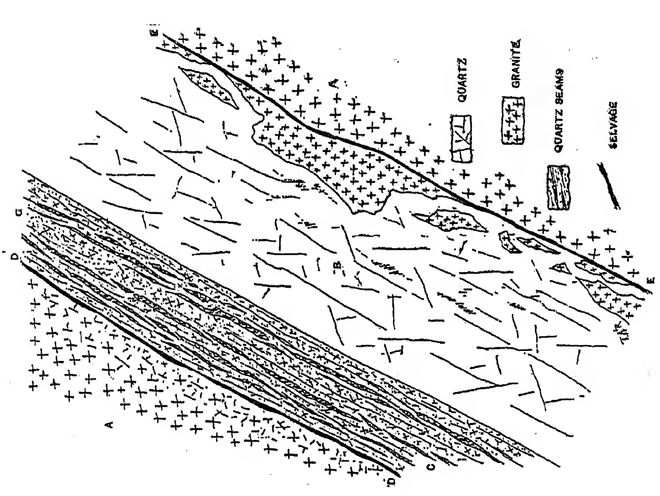
Illustrations of these conditions may be seen in Figs. 1 and 2. The first is reproduced from a sketch made June 25, 1895, in the lower level of the Union and Companion mine at Cornucopia, Union county, Oregon. It represents the breast of the north drift on the west vein. The country, a fine-grained granite, is not visibly altered under the foot-wall; but along the hanging it exhibits an alteration of its more soluble ingredients. There is a slight selvage, D, separating the granite from the pay-ore, C, which is about 10 inches thick, and consists of ribbons of quartz, impregnated with pyrite and alternating with strips of altered country. A distinct parting, unaccompanied by any apparent selvage, divides this streak of ore from one, B, below it, which is twice as thick, but much less gold-bearing. This part, B, of the vein consists of white quartz, carrying occasional patches of pyrite, and marked by large inclusions of slightly altered country, arranged along the foot-wall, where a thin selvage separates them from the outer granite. The evidence of vein-structure embodied in this figure permits diverse interpretations. The upper pay-streak, C, appears to me to be country-rock, in place, decomposed, fractured, and silicified, with accompanying precipitation of gold. The central wall may have been the original hanging-wall. The present foot-wall is sufficiently distinct; but the occurrence of the pieces of enclosed country leads me to believe that at an earlier stage

Fig. 2.



HILLSIDE MINE, ARIZONA.

Fig. I.



UNION AND COMPANION MINE, OREGON.

the foot-wall was broken and irregular; the shape and position of the fragments of rock now lying upon it being such as to render it doubtful that they could have been detached from the hanging.

Fig. 2* was drawn May 10, 1893, in the No. 4 level, north, of the Hillside mine, Yavapai county, Arizona. The lode occupies a strong fissure, cutting almost vertically through the nearly horizontal layers of a quartzose talc schist, B B. The original line of fracturing is probably now occupied by the seam, C, 6 inches thick, of white talcose clay, covering the foot-wall. The ore-bearing portion, D, of the lode is formed by an irregular mineralization of the hanging-wall country, extending to a distance of from 15 to 18 inches, and presents an intricate medley of quartz, pyrite, zinc-blende and a little galena, carrying about 1 ounce of gold and 25 ounces of silver per ton.

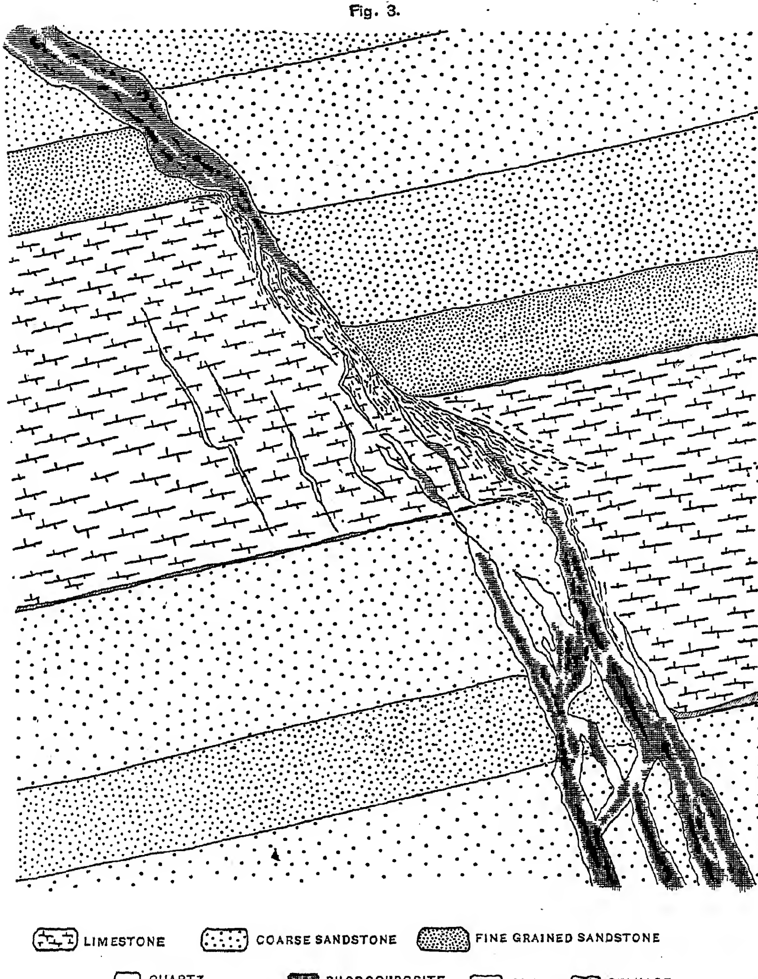
The most noteworthy feature of the section is the occurrence in the hanging, on the outer confines of the main ore-streak, of several irregular cavities, A, A, whose inner surface is covered by a series of siliceous coatings, evidently deposited by mineralizing waters that have circulated through them. Along the outcrop of the lode, at Wikiup Point, there occur hollows in the schists, of a character similar to those above described, and of such a shape as to suggest that their origin was due to the removal, by waters carrying carbonic acid, of certain portions of the country, rendered soluble by the segregation of lime. As the fourth level nearly follows the water-level of the mine, and the siliceous encrustations were stained with iron oxide, the formation appears to have been due to what Posepny called the vadose circulation. On the other hand, the impregnation of the hanging-wall country by sulphides cannot be ascribed to oxidizing waters, and must have taken place at an earlier period, when the surface was relatively more distant.

The lode follows a fissure formed along the axis of a synclinal bend in the schists, and often very noticeably reproduces the structure of the country which it has in part replaced; the ore breaking along lines corresponding to the almost horizontal foliation of the schists. The width of the ore is very irregular. That occasionally found under the clay seam is rarely rich enough to mine; the main pay-streak being that portion

* See also *Trans.*, xxiv., 945.

of the vein bounded underneath by the clay and extending into the hanging until the mineralization becomes so meager that "ore" becomes "country-rock."

Fig. 3.



ENTERPRISE MINE, COLORADO.

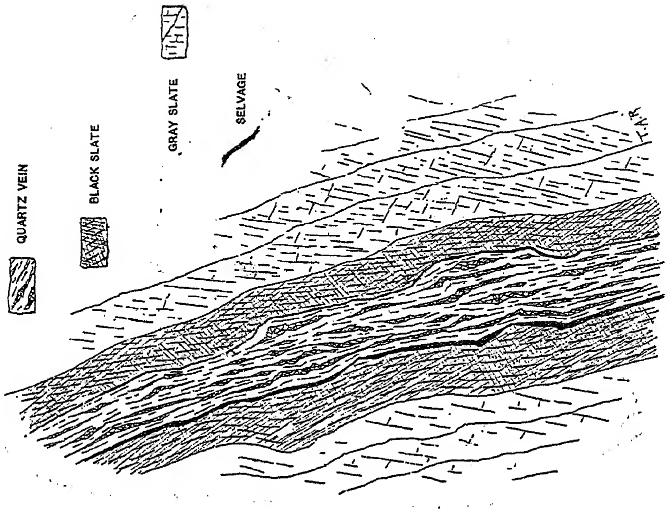
When a vein occurs in a formation composed of several kinds of rock it may cut across the lines of parting and be labelled a "true fissure;" or it may conform to them, and become a

"bedded vein," if the two beds happen to be similar, or a "contact-vein," if they are dissimilar. It is evident that, when a vein crosses the bedding of a series of sedimentary rocks, the differences between the enclosing walls at any given place will depend upon the thickness of the beds traversed, and the extent of the faulting of the country along the line of the fissure. When the faulting is slight, the change in the wall-rock will be practically simultaneous for both sides of the vein; while, when the dislocation is equal to, or exceeds, the thickness of the members of a series of dissimilar beds so intersected, the opposing walls may be entirely dissimilar. This is illustrated in Figs. 3 and 4.

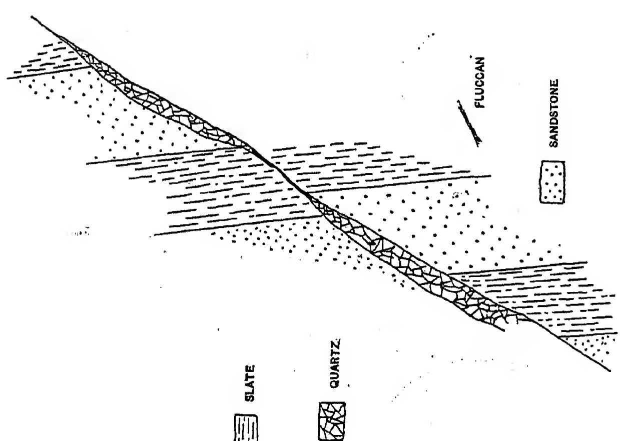
Fig. 3 represents the breast, on August 14, 1894, of the north drift of the Jumbo No. 2 vein, on the Group tunnel level, in the Enterprise mine, at Rico, Dolores county, Colorado. The vein follows a fault-fissure through a series of Lower Carboniferous shales, limestones and sandstones. The throw of the fault, along which the ore has been deposited, is about 2 feet; the thickness of the prominent bed of limestone is 3 feet; and the section shown in the figure covers 7 feet by 6. It is characteristic of the veins in this mine that they split up and become impoverished in lime, while in the sandstone, on the contrary, they usually become clean-cut, compact and richly ore-bearing, as is the case at the top of the drift represented in the figure. In traversing the lime, the selvage following the line of fissuring is very noticeable; but in the sandstone, particularly where the vein splits, the ore is "frozen," that is, has no evident parting separating it from the encasing rock.

Fig. 4 is taken from a drawing accompanying a note by Mr. E. J. Dunn, of the Victorian mining department, contributed by him to the Quarterly Report of December 31, 1888. It represents certain features of the Sunday reef, near Beechworth in Victoria (Australia). The country consists of Silurian slates and sandstones, which have been faulted about 2 feet. Along this line of faulting gold-bearing quartz has been deposited; and it is noticeable that its occurrence is mainly confined to the under side of the sandstone, while under the slate it disappears and gives place to fluccan or clay. I would suggest that the lenticular shape of the quartz-bodies indicates that the spaces

Fig. 5.



BONANZA VEIN, OREGON.



SUNDAY REEF, VICTORIA

occupied by them were produced by the movement of one of the walls of a fissure, following a line whose undulatory form was caused by the unequal texture and hardness of the beds traversed by it.

Of the change observable in the character and value of the mineral ingredients of a vein in its passage from one kind of rock into another it is hardly possible to speak in parenthesis. One of the best known examples is that of the old Dolcoath mine in Cornwall, where the vein, in leaving the clay-slate (killas) and penetrating the granite, changed from a copper-bearing into a tin-bearing lode. I might also mention the silver-lead veins of Pontgibaud,* in France, which are in a gneiss country, diversified by dikes of granulite. The ore-veins have been formed along fractures within the dikes, and on their line of contact with the gneiss. When the dike diminishes in size, the ore decreases in width; when the vein penetrates into the gneiss, the ore disappears. The best ore is associated with the kaolinization of the feldspar of the granulite; and when the latter becomes hard and unaltered in depth, the ore pinches out.

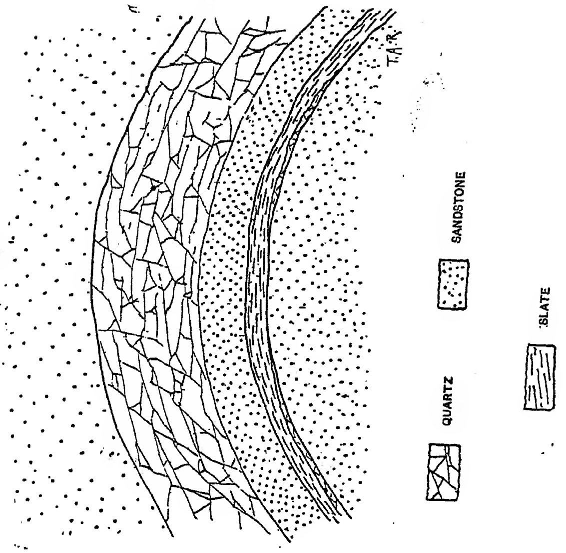
On Newman Hill, Rico, Colorado, the veins of rich gold- and silver-bearing ores are noticeably affected by the character of their rock-walls. The particular changes due to penetrating from lime into sandstone have already been mentioned in connection with the veins of the Enterprise mine, but there is also the more general observation, that when the sedimentary beds are black, the veins in them are rich; when they lose that black color, the ore diminishes.

Other instances occur to me, but the above are typical. This inter-dependence between country and ore has been used as an argument in support of the now crippled lateral-secretion theory. It has been suggested† that this relation, often noticed in vein-mining, points to the derivation of the ore from the enclosing rock, and that some formations have an enriching effect, because they have been the source of the valuable metals

* See "The Lodes of Pontgibaud," by the writer, in the *Eng and Min. Jour.* of August 11 and 18, 1894.

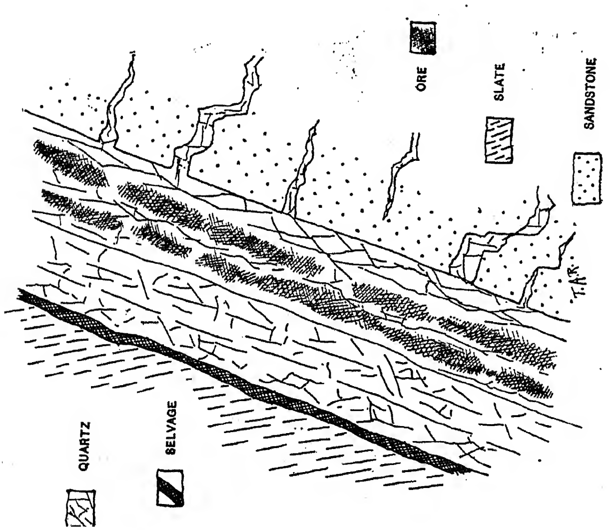
† As, for instance in the paper "On Some Evidences of the Formation of Ore-Deposits by Lateral Secretion in the John Jay mine, at Providence, Boulder county, Colo.," by P. H. Van Diest, in the *Proceedings of the Colorado Scientific Society*, vol. iv., p. 340, and in the discussion of the said paper, *Id.*, p. 340.

Fig. 7



JOHNSON'S MINE, VICTORIA.

Fig. 6.



SHENANDOAH MINE, VICTORIA.

now found in the veins penetrating them. But as Cotta long ago suggested, the influence of the physical texture and chemical composition of the country, as facilitating the deposition of the ore, may explain this phenomenon. The former would affect the rate of cooling and the formation of adhesive crusts. The latter would act by direct chemical precipitation.

As I suggested in the discussion of the paper just referred to, the local enrichment or impoverishment of veins may be explained by the presence or absence in the enclosing formation of precipitating agents. What the agent has been we can only in rare instances guess. At Rico it was undoubtedly the carbonaceous matter enclosed in the Lower Carboniferous shales, limestones and sandstones. At Pontgibaud it was probably the feldspar which made room for the silver-bearing galena, and in Cornwall also the beautiful pseudomorphs of tinstone after feldspar suggest similar chemical interchanges.

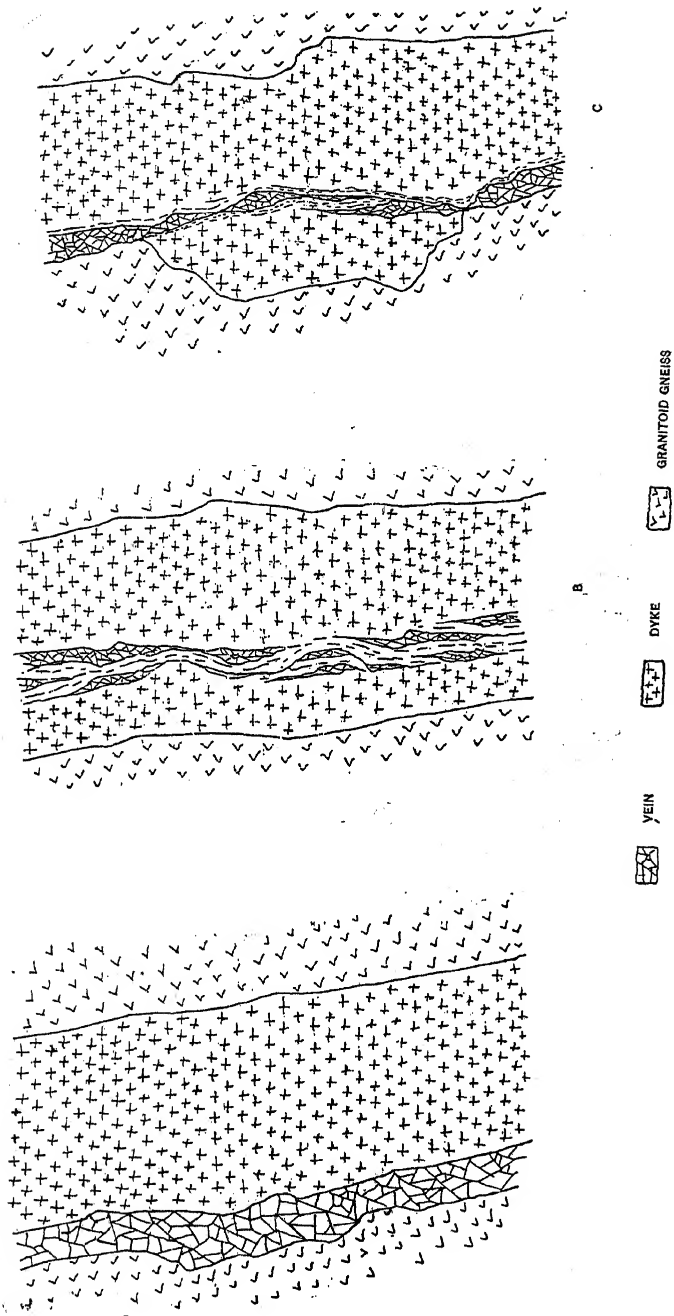
In the case of veins which lie along the bedding-planes of sedimentary rocks, the dissimilarity between the enclosing walls may not go further than a slight difference in the grain of two beds of sandstone, the color of two beds of slate, etc., or it may reach the more marked diversity presented by rocks as entirely unlike as a quartzitic sandstone and a soft slate.

Fig. 5 represents a gold-vein, following the bedding of, and encased by, a band of black slate, which is in turn flanked on either side by light gray slates. The ore consists of ribbons of quartz, mingled with strips of included country, and separated from the outer slates by a selvage, faint on the hanging- but strong on the foot-wall. The drawing was made July 3, 1895, in the upper level of the Bonanza mine, Baker county, Oregon.

The comparatively straight walls of ordinary vein-mining occasionally give place in veins of the bedded class to surfaces having a marked curvature. Such walls characterize the saddle-reef, a type of lode-structure common in only two known mining districts, namely, Bendigo in Australia and Waverly in Nova Scotia—unless it be true, as is now stated on good authority, that the Broken Hill lode in New South Wales is also a saddle-reef.

In these regions, gold-bearing quartz is found along the bedding-planes of folded sedimentary rocks. While anticlinal folds (or saddles) alternate with synclines (inverted saddles or

Fig. 8



TYPES OF VEIN STRUCTURE IN GILPIN COUNTY COLORADO.

troughs), experience has shown that the ore-deposition is mainly confined to the former. Such a formation will offer many striking features, because of the occasionally very regular curvature of the walls. I remember, for instance, standing in the stopes just above the 980-foot level in the Johnson's mine at Bendigo, and seeing the foot-wall curve underneath like the top of a boiler, while the hanging arched overhead like a Roman bridge. This was the apex of a saddle, as illustrated in Fig. 7, reproduced from a sketch made at the time.* The lode is seen to consist of white quartz about 2½ feet thick, separated from the overlying sandstone by a very regular parting of black clay. Underneath is about a foot of sandstone, then a dark seam of slate, from 5 to 6 inches thick, whose parting from the next bed of sandstone is marked by streaks of quartz, thinning out both east and west.

The downward continuation of such a formation (the "legs of the saddle") presents the appearance of an ordinary bedded vein, usually marked, however, by a noteworthy want of persistence of ore in depth. Of the many drawings illustrating such veins already contributed to the *Transactions*,† I have reproduced, in Fig. 6, the breast of the north end of the 1990-foot level in the Shenandoah mine at Bendigo. The lode carries 2 feet of closely-laminated quartz, from which spurs or stringers go off into the underlying sandstone. The hanging shows a gouge or selvage,‡ separating the quartz from the overlying slate.

Many veins follow the contact between eruptive dikes and the metamorphic or sedimentary formations which they have penetrated. The dikes of quartz-andesite porphyry traversing the granitoid gneiss of the earliest mining districts of Colorado (in Boulder, Gilpin and Clear Creek counties) offer many examples of this type of vein-structure. In such cases the mineralization may often be found to have spent itself on the more

* October 5, 1890. See also *Trans.*, xx., 506.

† By the writer in vols. xx. and xxi.

‡ "Selvage," "gouge," "dig," "pug," "fluccan" are all more or less synonymous. "A layer of soft stuff" would cover them all. It is perhaps worthy of notice, however, that our "selvage," used in this sense, is not the exact synonym, as it has often been supposed to be, of the German *Saalband*. A *Saalband* is a definite wall, as distinguished from a gradual transition from vein matter into country-rock. A layer of soft material on the wall is a *Besteg*.

Fig. 10.

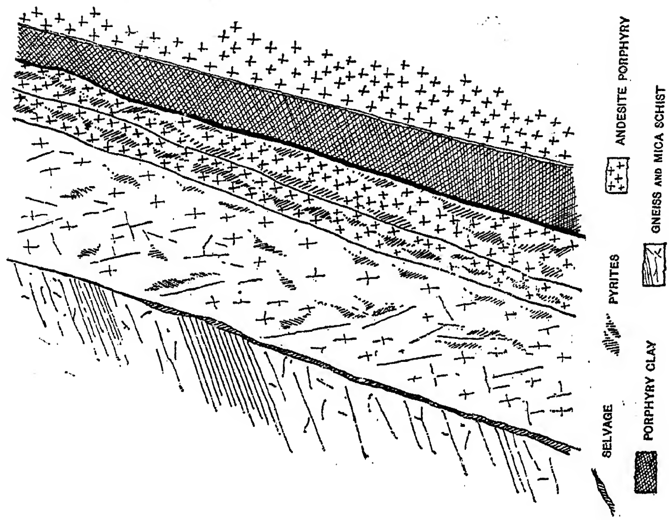
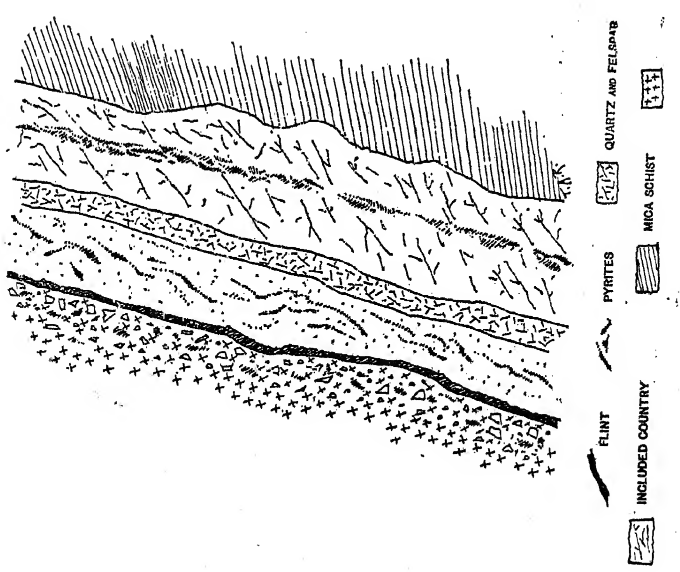


Fig. 9



CALIFORNIA MINE, COLORADO.

soluble porphyritic igneous rock, rather than upon the less soluble metamorphic. The walls of such veins will vary, as the ore deposition has followed either fractures along the immediate contact, or those which ramify into the body of the dike, or those again which cut across the latter, where its irregular outline has been an obstacle to the main line of fissuring. These ideas are illustrated in the diagrams A, B and C, Fig. 8.

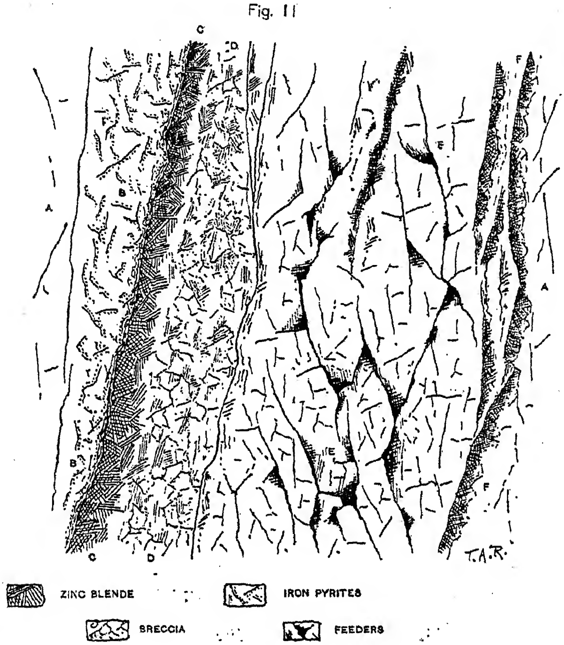
The California mine, in Gilpin county, offers many examples of such vein-phenomena. Figs. 9 and 10 represent the western ends of the 2000-foot and the 2100-foot levels, as seen on July 13, 1892. In the first the vein is seen to lie between mica-schist, on the foot, and "porphyry," on the hanging. The "porphyry" forms part of a dike, 17 feet thick, of dacite or quartz-andesite, and is both brecciated and much decomposed near the lode, from which it is separated by a dark band of "flint," which consists of small fragments of porphyry cemented together by a very dark chalcedonic quartz. Underneath this there are 5 inches of white kaolinized porphyry, containing threads of iron and copper pyrites. Next comes an inch and a-half of quartz and feldspar intermingled; then a band of included country, part gneiss and part mica-schist, which is subdivided by a streak of pyrite. Finally there is an irregular foot-wall; the lode-filling shading off into the soft mica-schist which underlies the vein.

The lower level, shown in Fig. 10, exhibits a marked difference. The lode has crossed the dike, and the porphyry forms the foot-wall. Next comes a thickness of 6 to 8 inches of white, soft, decomposed porphyry, then a black selvage, with slickensides on the lower side. Then come two bands of mineralized porphyry, separated by thin partings. The main width of ore consists of about 2 feet of lode-filling traversed by patches and streaks of pyrite. Fragments of porphyry can also be recognized in it. This is separated from the overhanging gneiss and mica-schist by a selvage of varying thickness.

In the neighboring Indiana claim, the California vein exhibits certain changes, the most evident of which are the absence of selvages, the indistinctness of its limits and the brecciation of the vein-filling. This is suggested in Fig. 11, which represents the breast of a stope above the 800-foot level west, as observed November 13, 1895. The enclosing country, A, A, is a

granite almost destitute of mica. The part B is bespattered with pyrite. The best ore is a seam, C C, of black zinc-blende lining the hanging-wall. D is evidently brecciated. The larger part of the section consists of slightly altered country (E E) reticulated with seams of blende, following joint-fractures. The foot-wall of the vein is considered to be under the bands of zinc-blende and copper pyrites occurring along F F. The en-

Fig. 11



tire width is about 4 feet. The lode has departed from the dike, with which it is so closely associated in the neighboring mine; but the workings show that it meets this dike at intervals, and is benefited by the intersection.

That the vein follows the line of a fault can be seen by examining the walls of the 2000-foot level in the California mine, more particularly at points between 350 and 450 feet west of the shaft, where the lode has left the dike entirely, and is encased in the gneiss and mica-schist. The country-rock on the two sides of the drift is not the same. The extent of the throw of the fault, however, could not be measured.

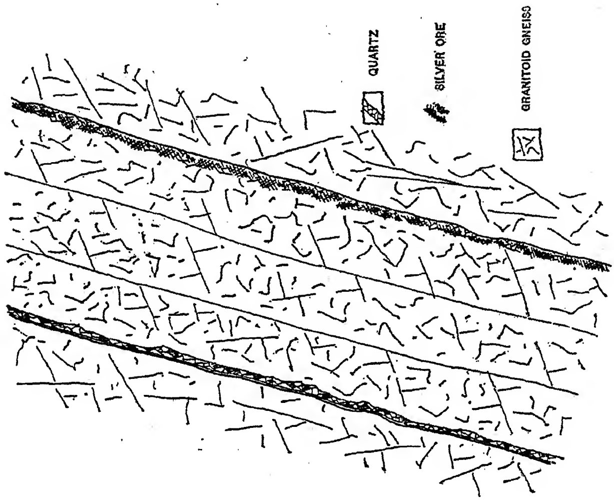
In the course of the foregoing descriptions of lode-structures, mention has been repeatedly made of the occurrence of clay selvage, following sometimes one, sometimes both, of the walls of a vein. This "clay" may occasionally be material precipitated from solution; ordinarily it is only crushed rock. It frequently encloses exquisite mineral specimens, because its soft consistency has permitted untrammeled crystalline growth. Most examples of well-developed crystals of native gold have been discovered under such conditions. This is the case at Cripple Creek, Colo., where the gouge or clay has been dried and hardened near the surface, and as a crumbly earth, made purple by the presence of fluorite, carries beautiful crystals of gold pseudomorphic after sylvanite and calaverite. The exquisite leaf-gold specimens, for which Farncomb hill (Breckenridge, Summit county, Colo.) is so famous, are found imbedded in talcose clay. Large pieces of pure argentite are often found in such an environment, as at the De Lamar mine, in Owyhee county, Idaho. Wire-silver also has been found in comparatively large amount encased in such a "mud" in many Leadville mines; notably at the Crown Point, in 1886.

By reason of their opposition to the passage of water such seams of clay protect the rock-surface of vein-walls, and underneath them there will occasionally be found comparatively fresh and unaltered rock having beautifully polished faces or slickensides. At Ballarat, in Australia, I have seen many such rock-faces like finished ivory in their smoothness, and streaked with black lines, due to the grinding of specks of pyrite. In the Bonanza mine, Baker county, Ore., there could be seen quite recently an exquisite example of such an occurrence. In an upper drift there was at one place a surface of a few feet square (on one of the walls of a gold-bearing quartz-vein) covered by a thin layer of black clay, under which lay what seemed a white enamel of very remarkable delicacy. It could not be removed without breaking, because it was very friable, consisting essentially of crushed quartz partially recemented, probably by pressure.

"The handwriting on the wall" is not always easy to decipher. The lines or striæ occasionally to be seen upon its surface have been held to indicate the direction of that movement (or succession of movements) of the opposing rock-planes to

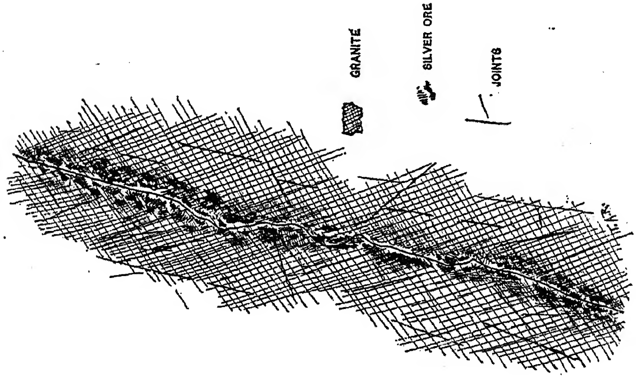
which the deposit of ore primarily owed the opportunity for its existence. These lines, however, sometimes have opposite di-

Fig. 13



SEVEN-THIRTY MINE, COLORADO.

Fig. 12



rections within a short distance and offer conflicting evidence hard to explain.

Rarely is a story told more clearly than in the ripple-marked foot-wall which was to be seen in October, 1891, in the Johnson's mine at Bendigo. It had been very difficult to distinguish the bedding of the country, because the development of a strongly-marked cleavage had obliterated the lines of original sedimentation. At the 1065-foot level, however, the matter was made plain. For more than 100 feet square the surface of the foot-wall was covered with ripple-markings. The crests of the waves were about 3 inches apart and presented all the little irregularities to be seen to-day when the wind blows over the shallow waters of an estuary and imprints the evidence of its action upon the yielding sand. The markings had been protected by layers of Silurian sediment, and the whole series had been indurated into rock, the sand which bore the markings becoming quartzitic sandstone, and the overlying mud slate. Between them, as within the pages of a book, was preserved the conclusive evidence of the original position of the beds of rock enclosing the reef, which had been formed in later times, when fissuring had made room for the circulation of underground waters and the deposition of the gold-bearing quartz.

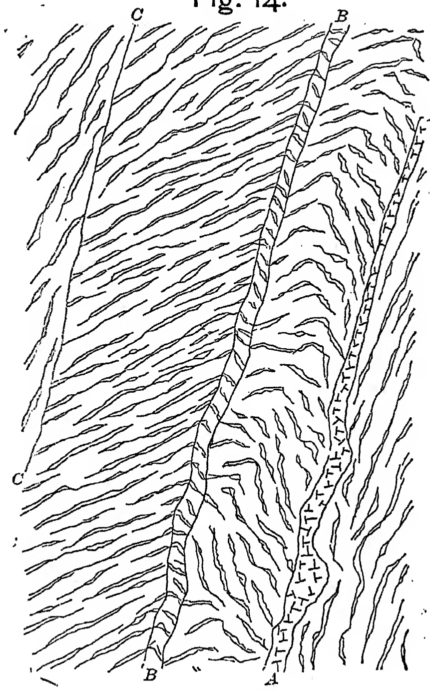
In the above interesting case the corrugation of the foot-wall, due to the ripple-markings, rendered difficult the detachment of the ore. Distinct walls, especially when accompanied by selvage, are very useful in actual mining; but they are not by any means necessarily indicative of a productive vein, or particularly favorable to the continuity of the ore. A "clean" wall and a good "gouge" are welcomed by the miner because they ease his toil; but the idea that their presence alongside a lode gives it a character better than another unprovided with such adjuncts is a dangerous delusion. In many mines, more ore has been lost through the persistent following of a "wall," without exploring beyond it, than was ever compensated for by the greater facility given by such a parting-plane for the breaking of the ore found.

Many veins have no defined walls, but gradate imperceptibly into the enclosing country, and are bounded only by the commercial value of the material mined. Such veins are to be seen, for instance, in the mountains that overlook Silver Plume, Clear Creek county, Colo. Fig. 12 represents a sketch made May 27, 1892, from the 300-foot level of the Seven-Thirty mine.

A fracture penetrating the metamorphic granite carries ore on both sides, which diminishes in richness as it spreads into the encasing country. The joints in the granite are evident.

In this mine the so-called "walls" are often simply two parallel veins (rich, but very small), separated by clean, hard country. This is illustrated in Fig. 13, which was obtained

Fig. 14.



CANTON MINE

from the same level about 1000 feet further east. The granitoid gneiss is traversed by two streaks of ore, of which the one to the right is much the richer. Between them there are at least two well-marked parallel fractures devoid of ore. The vein to the left has a thin selvage, under which there is a streak of quartz carrying a little silver-ore; but the companion-vein to the right follows a fracture, unaccompanied by any selvage, whose upper side is impregnated with about 3 inches of tetrahedrite, galena, and polybasite.

Where ore is absent in the Seven-Thirty mine, the walls are apt to be particularly well-defined; and when there is any thickness of rich silver-bearing mineral present, the walls are scarcely to be distinguished, and the rock is hard to break, because it is destitute of convenient partings. The large veins carrying gouge are found to be uniformly poor, except where they meet the very narrow rich streaks which constitute the resource of the property. The Seven-Thirty vein proper is only 2½ inches thick, but it is very persistent through the midst of hard crystalline rocks, and it has, for twenty years, proved very productive.

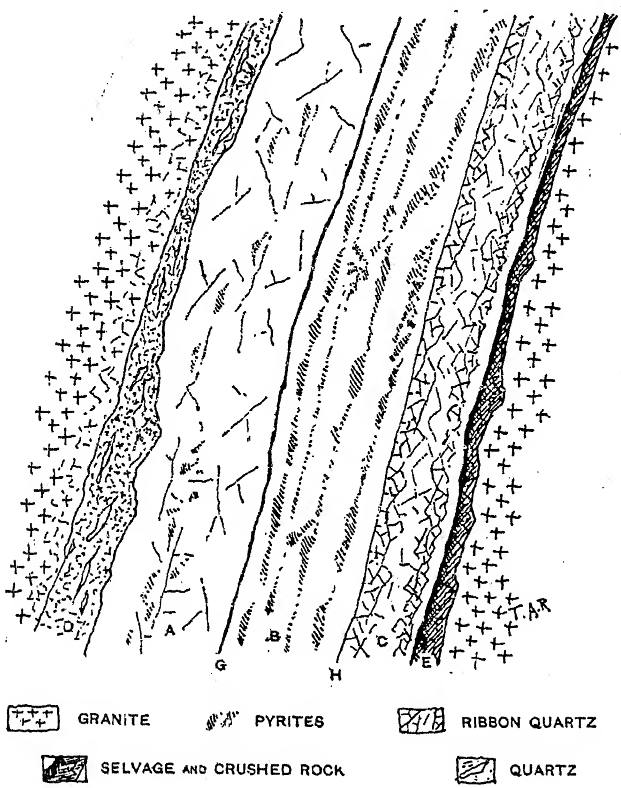
In many mines one vein only is exploited, and cross-cutting the country in search for parallel lodes is entirely neglected. In others, a cross-cut is stopped as soon as it reaches the further wall of the particular vein it was started to reach. Both these unwise practices are founded upon a misconception of lode-structure, due to a narrow interpretation of the early teachings of economic geology, which lays a misleading emphasis upon the definition and clean-cut boundaries of so-called "true fissure-veins." The fact is, as daily observation proves, that there are walls within walls, and walls beyond walls; and that to follow closely any particular hard, smooth rock-surface, with the idea that it is the utmost limit of ore-occurrence in any particular mine, is to be blind to the realities of geological structure.

Fig. 14 represents the face of a drift* in the Canton mine, near Waipori, Otago, New Zealand. A A is the reef, a vein of quartz which is supposed to lie immediately upon the foot-wall. Along B B the quartzose schist is soft, and the included quartz-folia are much twisted. C C is one of the so-called "false hanging-walls." Along A A and C C faulting is evident, along B B distortion only. It was not possible to say where the lode ended, or where it began. The whole width from A to C was known to be gold-bearing, although A A served as a guide in following the gold-bearing channel. Nevertheless those who were working the mine had little comprehension of the formation, particularly of its essential lack of definition, and, while admitting that there were several "false hanging-walls," insisted that there was only one foot-wall (underneath A A) which was stated to be of a different

* On November 15, 1890. See also *Trans.*, xxi., 415.

kind of rock, and exceptionally hard. On examination I found that the rock of the supposed foot-wall was similar to that of the rest of the gold-bearing country forming the lode, and on a sample of it being crushed and tested in a prospector's

Fig. 15.



UNION AND COMPANION MINE, OREGON.

pan, it was discovered to be richer than that which was being actually mined. It was scarcely necessary after that to insist that a cross-cut should be made into the foot-wall.

Fig. 15 represents the north breast* of the lower level on the main lode in the Union and Companion mine, Union

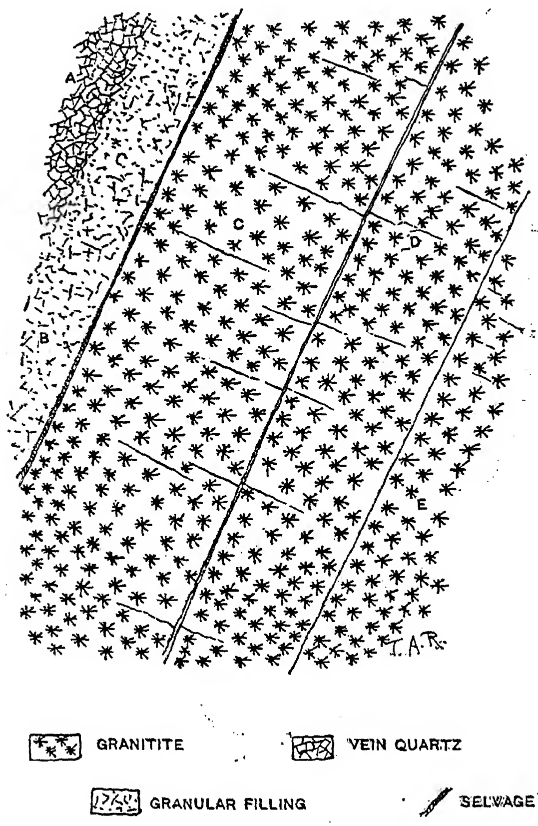
* On June 26, 1895.

county, Oregon. It illustrates the occurrence of "walls within walls," for while the lode may be limited by the main boundaries along E and D, there are at least two partings (G and H) equally well-defined, subdividing the enclosed width of ore. The country is a fine-grained granite, which, near the hanging, is decomposed and ore-bearing. D is a streak of granular crushed country, mixed with lenticles of white quartz whose longer axes are parallel to the lode-walls. D is from 3 to 7 inches wide, and carries only traces of gold. A consists of white hackly quartz spotted with iron pyrites. It is from 14 inches to 2 feet wide, and contains about ½ an ounce of gold per ton of ore. Then comes a hard regular "wall," separating A from B, which is the main pay-streak, ribboned with veins of iron and copper pyrites. The width is from 2½ to 3 feet, and the ore averages about 2 ounces in gold and 8 ounces in silver. Then follows a parting marked by a slight selvage, underneath which comes a 10- to 15-inch band (C) of ribboned white quartz, stained by the oxidation of copper pyrites and carrying about 5 pennyweights of gold per ton. Then comes the main foot-wall with its streak, 1 to 3 inches thick, of granular crushed country, mixed with clay. The underlying rock is but little altered.

Fig. 16 affords an example of "walls beyond walls." It represents a section obtained at the station on the 500-foot level in the Mammoth mine, Pinal county, Ariz. The Mammoth lode traverses hornblende-granitite, porphyrite and a porphyry agglomerate. The lode-filling consists of altered country, and therefore changes as the lode in its strike penetrates first one kind of rock and then another. When standing in the stopes it is not difficult to recognize in the ore the reproduction of the habit of either the granite or the porphyrite by whose alteration the lode was produced. The country near the lode is much altered and often visibly gradates into the ore, while, as the lode is receded from, these effects diminish until they become confined to the faces of the rock lining the fractures. The granitite carries two feldspars, of which the pink orthoclase is evidently more stable and succumbs to decomposition less quickly than the green plagioclase. The ore is both gold- and silver-bearing, but chiefly valuable for gold. The great variety of associated minerals includes some uncommon species, such as wulfenite (usually colored by vanadic acid), vanadinite, des-

cloizite, ecdemite, dechenite, linarite, besides the commoner anglesite, pyromorphite, cerussite, malachite, dioptase, azurite, and a little galena and pyrite. Referring to the drawing (made March 17, 1893), the edge of the ore is shown at A; it becomes

Fig. 16.



MAMMOTH MINE, ARIZONA.

mixed with altered granular country (along B) in approaching the "main foot-wall." This is followed by the granitite itself, in which there are well-defined walls (or fractures parallel to the lode-channel) and cross-joints, often lined with exquisite crystals of vanadinite and wulfenite.

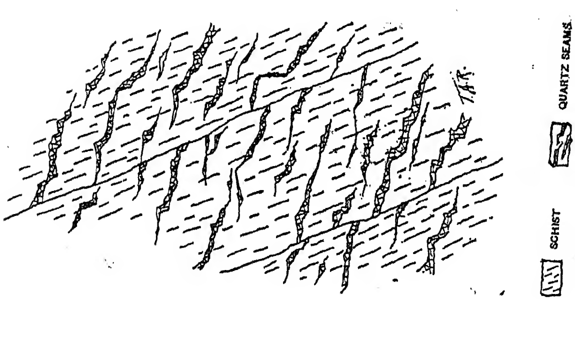
Going to the Pacific coast, Fig. 17 represents a part of the west side of the so-called "mother lode" of California. The

drawing, made May 21, 1891, is a portion of the face of a large open cut at the Gold Cliff mine, Angel's Camp, Calaveras county, Cal., near the now well-known Utica mine. The ore-channel consists of a country-rock traversed by cross-veins of white gold-bearing quartz. The country-rock is a greenish-gray augite schist (probably at one time a diabase), carrying coarse pyrite near the gold-quartz.

There are "walls" *ad infinitum*. Each cuts off the quartz-seams, which occur again on the further side and extend to the next "wall," where they are terminated as before, and so on. A certain portion, 20 to 30 feet in width, of this channel of country is rich enough to work, and is sent to the mill, but the poorer material which lies beyond it has an identical geological structure. Of course, in such a case the "main walls" will depend for their determination upon commercial rather than geological conditions.

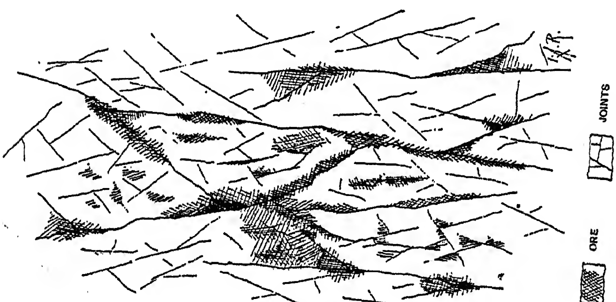
Another case in point is presented at the Cashier mine in Summit county, Colo., as illustrated in Fig. 18, which shows a part of an open cut on the lode, as seen August 22, 1895. The latter consists of altered quartz-felsite, rendered porphyritic by large crystals of feldspar. It is spoken of as a vein 45 feet wide, having a hanging-wall of porphyry and a foot-wall of lime. The ore is said to be penetrated by dikes of porphyry. The facts are really these: A certain width of quartz felsite within the neighborhood of its contact with the limestone has been acted upon by mineral solutions which probably came up along that contact. There are no walls, the porphyry of the hanging being simply the rock of the ore-channel in a less altered condition. The feldspar of the lode-rock has been leached out. In the cavities, now partially filled with crystalline quartz, iron oxide and gold, there can be distinguished the outlines of the large (½ to 1½ inches) crystals of orthoclase whose removal made the rock porous to circulating waters. The mineralization is indicated by the softening and reddening of the porphyry and is most marked along the joints, especially where they intersect. There are occasional portions of the rock comparatively unaffected by the leaching agencies, and therefore appearing as hard, unstained nuclei amid a mass of softer reddish ore. It is these that are locally termed "horses" and "dikes" of porphyry.

Fig. 17.



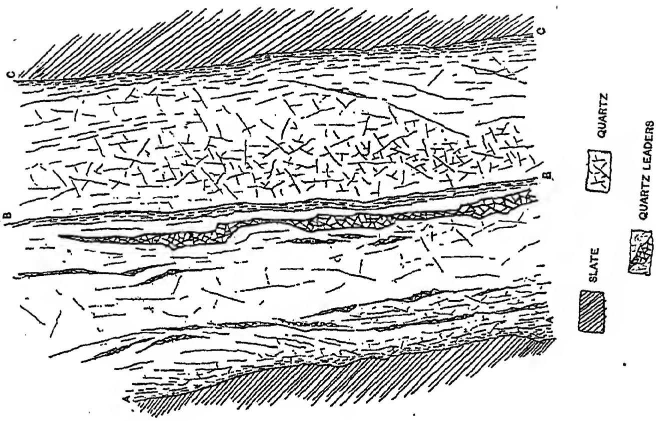
GOLD CLIFF MINE, CALIFORNIA.

Fig. 18.



CASHIER MINE, BRECKENRIDGE, COLORADO.

Fig. 19.



DRUMLUMMON MINE, MONTANA.

Lodes subdivided by partings parallel to their outer walls (as in numbers 1, 9, 10 and 12) often resemble twin veins such as are actually formed by the temporary parallelism of two distinct fissures travelling together after they have united. Such a case is shown in Fig. 19, which illustrates the union of the Old and New Castletown veins as seen in the north face of the 500-foot level of the Drumlummon mine, Marysville, Mont. The country is clay slate. A B is the Old Castletown vein, 2½ feet wide. B C is the New Castletown, 2 feet wide. There is no selvage on any one of the three walls, but each is marked by soft, crushed and foliated slate.

The generous lodes of silver-bearing copper-ore which at Butte, Mont., penetrate the granite are frequently marked by a brecciation of the encasing country and are accompanied by a mineralization of the granite far beyond the walls or limits of workable ore. In the 300-foot level of the Gagnon mine, 374 feet west of the main shaft, a cross-cut shows that the lode-channel extends 30 feet north of the supposed foot-wall, the enclosed granite being broken and mineralized. Beyond this line the country ceases to be shattered and is no longer impregnated with ore, but is comparatively fresh, hard, normal granite, with a blocky fracture. This outer foot-wall of the lode-channel is marked by the occurrence of some ore-streaks and an accompaniment of seams of clay, as is shown in Fig. 20 (drawn September 15, 1895). The foot-wall country has a noticeable number of slips or joint-planes. It is separated from the lode-channel by a thick layer of tough black clay. Then comes a zone of kaolinized granitic filling traversed by irregular veins of zinc-blende and pyrite. Another clay-seam divides this part of the section from a band of mixed white quartz and granitic filling, followed by altered mineralized granite, ribboned with veins of gray quartz, whose southern limit is a third seam of black clay. Then comes crushed, brecciated granite, diversified by quartz and occasional evidences of ore, which extends to the main pay-vein (on the hanging) which has been the workable part of the deposit. The section in the figure represents a width of 6 feet.

Fig. 21 came from the east breast of the 1300-foot level, in the same mine. It is a representative section of the main ore-bearing vein. Granite, visibly altered, marks the northern

Fig. 21.

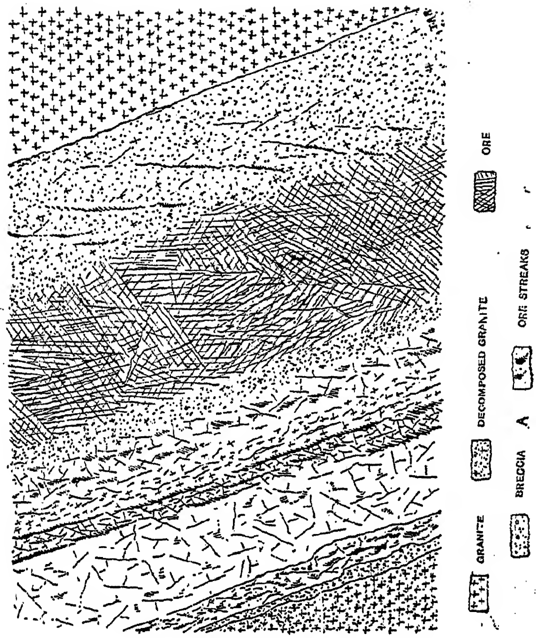
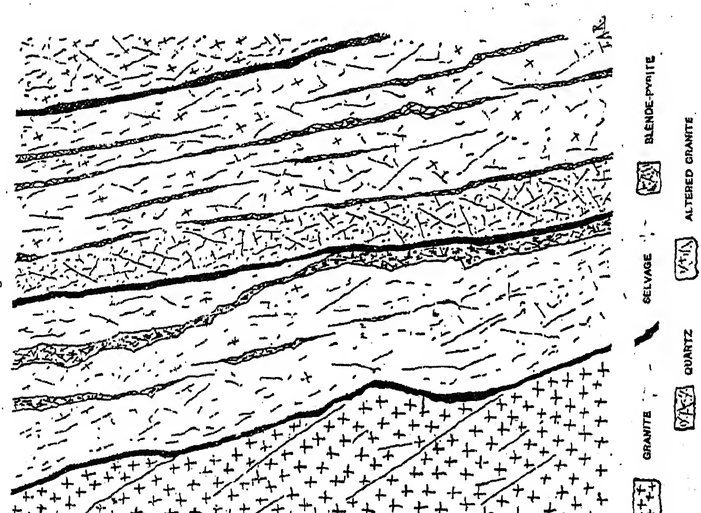


Fig. 20



THE GAGNON MINE, BUTTE CITY, MONTANA.

edge of the section, which is the foot-wall. Upon it lie a few inches of breccia, succeeded by 8 inches of blende, pyrite, and enargite, well intermingled. Then come 3 feet of friable, mineralized, light gray shattered quartz, giving place to 6 to 8 inches of harder quartz streaked with veinlets of chalcopyrite and bornite. Upon this lies a foot of altered granite traversed by streaks of quartz. Then come 18 inches of low-grade ore, consisting of quartz, pyrite, blende and a little bornite, separated from the main pay-streak by 6 inches of granitic filling. The main pay-streak is from 5½ to 6 feet wide, and consists of a massive mixture of pyrite, blende, enargite and bornite, carrying about 60 per cent. of silica. Between this and the hanging-wall there is 2 or 3 feet of decomposed broken granite, showing small veins of ore which drop into the main pay-streak. Beyond is granite.

In mines of this character, the geologist may determine the existence of the lode far beyond the limits of workable ore; but the miner will rightly distinguish between what is mineralized* country too poor to exploit and the concentrated mineral which will yield a profit.

That straight walls are not the necessary adjuncts of a vein of ore is suggested in Fig. 22, which represents the breast (on September 26, 1895) of the hanging-wall drift on the upper level of the Double Extension mine, in Summit county, Colo. The lode-formation consists of gently sloping quartzite, cut across and broken into by porphyry, which, as a dike, forms "the main vein," and in the shape of sheets, intercalated among the beds of quartzite, makes a succession of "floors" of gold-bearing ore of widely varying hardness. In the particular section illustrated, the intrusive porphyry forms the hanging-wall, A B C, of a zone of ore which is limited on its lower side by the ragged edges of the quartzite, M M. In this instance the conventional straight walls give place to one of extreme and irregular curvature and another of a markedly broken and

* The term "mineralized," like the word "mineral," is employed by miners in a sense not sanctioned by the ordinary dictionary, though fully entitled by its general usage to such recognition. Dr. Raymond's *Glossary of Mining and Metallurgical Terms* gives this sense as follows: "*Mineral.* In miners' parlance, ore. . . . *Mineralized.* Charged or impregnated with metalliferous mineral." The French use *minerai* and *mineraliser* in this sense; and I have adopted it because no English equivalent occurs to me.

jagged outline; but they are nevertheless walls, as truly as would be the most perfectly straight, smooth rock faces.

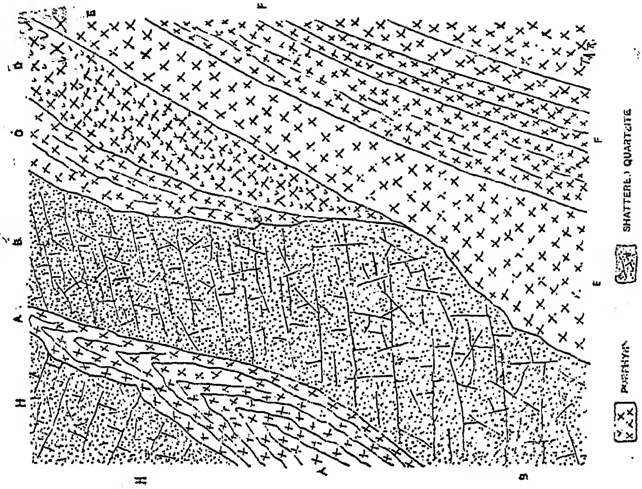


Fig. 23.

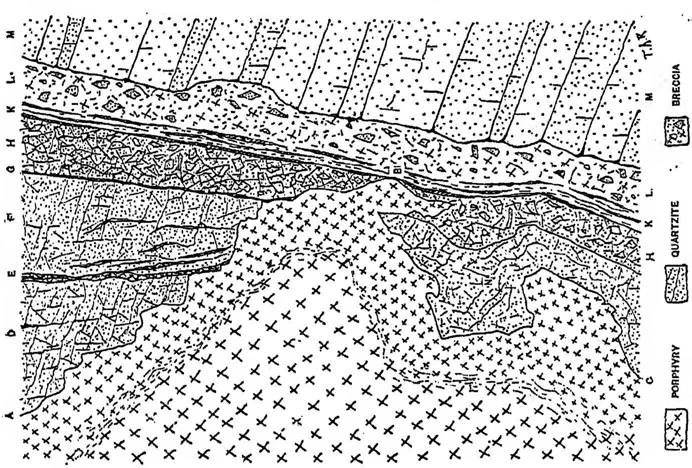


Fig. 22.

DOUBLE EXTENSION MINE, BRECKENRIDGE, COLORADO.

The porphyry, A C, a quartz-felsite, is bleached to a yellow white and softened to a granular clay, as it approaches its con-

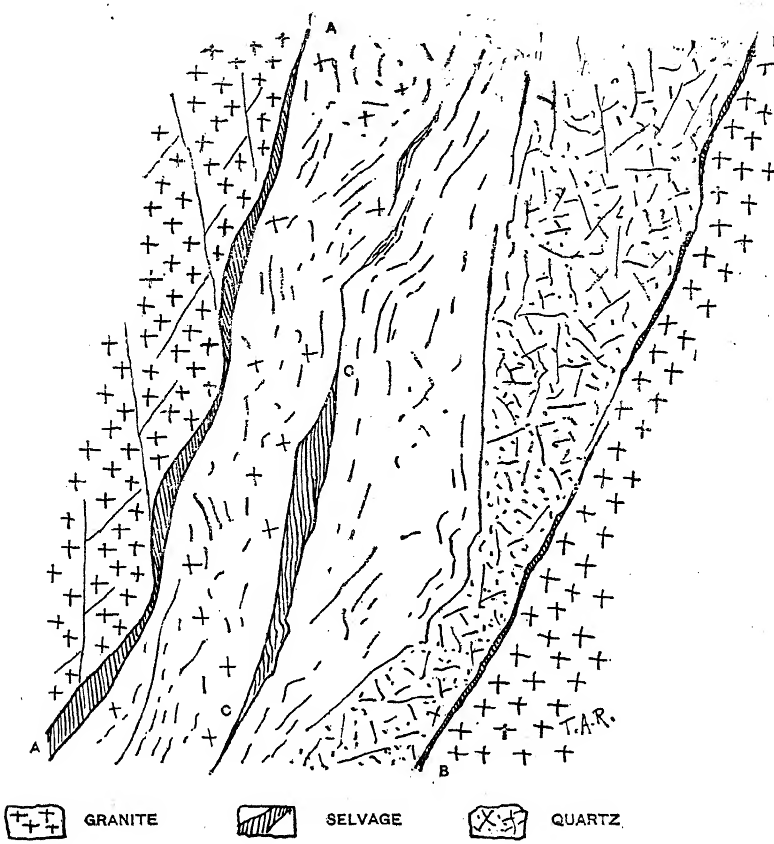
tact with the broken quartzite, D. The latter is dark bluish-gray, and carries, along its joints and other fractures, minute seams of iron-stained ocherous clay, which is gold-bearing. E and G are veins of such gold-bearing ocher. F is crushed quartzite, very similar to D. The band of quartzite breccia, H H, is separated from an equally wide band of porphyry-quartzite breccia, L L, by a succession of thin parallel quartz-seams, K K. Then comes the foot-wall itself. At N, under the projecting curve of the porphyry of the hanging, there is a mass of much-shattered quartzite mixed with iron-stained quartz. This is all gold-bearing.

Fig. 23, representing the western edge of a "cutting-out stope," near the supposed foot-wall of the lode (as seen September 23, 1895) exhibits a somewhat similar complication and another curved "wall." To the extreme left is fractured quartzite, carrying iron-stained clay along the faces of fractures, and divided into two parts, H H and B B, by a narrow zone, A A, of soft yellow porphyry, whose curved lines of alteration are marked by streaks of gold-bearing ocher. The remainder of the section is all porphyritic material, of which C C is similar to A A; D is a wedge of comparatively fresh rock, but slightly kaolinized and full of pyrite, and E E and F F are layers of brown and reddish soft talcose porphyry and clay, separated by numerous slips or smooth partings forming "false walls." The "main foot-wall" was supposed for some time to be the line of contact of the band of quartzite, B B, with the underlying porphyry; but assays have shown that the soft decomposed rock lying beyond it is fully as gold-bearing as the quartzite, and can be mined with as much profit for several feet beyond that line.

Not infrequently veins have irregular indistinct walls when ore-bearing, and smooth, clearly defined ones when barren. Fig. 24 illustrates the face (as seen September 15, 1895) of the west drift of the 430-foot level on the middle vein in the Nettie mine, near Butte City, Mont. The south country is a fairly hard, reddish granite, which, in approaching the hanging, becomes soft and is traversed by slips or joints. On the hanging-wall itself, A A, there is a seam of tough black clay, in which can be seen frequent films of minutely crystalline blende and galena, and small imbedded shots of ore and rock. This over-

lies a filling of white decomposed granitic material, full of partings and seams of black clay, such as C C. The foot-wall, B B, is also marked by a black selvage. Underneath it comes comparatively fresh "Blue-bird" granite.

Fig. 24



NETTIE MINE, MONTANA.

A few feet further east this level carried an ore-body, A B in Fig. 24 being the zone so transformed. The brecciated quartz upon the foot-wall was the part of the vein which first became ore. The sides of the drift are now coated with a delicate efflorescence of goslarite (sulphate of zinc). In this con-

nection it may be of interest to state that in the Gagnon mine, at Butte, three miles away, the apparently clean country, at some distance from the lode, was found* to carry 3 per cent. of zinc, indicating the extent to which the mineralizing action had penetrated.

Figs. 25 and 26 represent an interesting piece of evidence. Fig. 25 is an attempt to reproduce in color a block of ore giving a section of the Jumbo vein, and broken in the Enterprise mine a year ago. It is a beautiful example of ribbon-structure. The general lode arrangement is shown in Fig. 26, from a sketch made November 19, 1894. The vein follows a line of faulting, nearly at right angles across sedimentary beds of alternating sandstone and limestone. The extent of the fault is clearly marked by the dislocation of the bed of lime, B B, and its down-throw of about 2 feet on the hanging-wall. The country on the foot has its bedding-planes turned down, while on the hanging the reverse occurs.

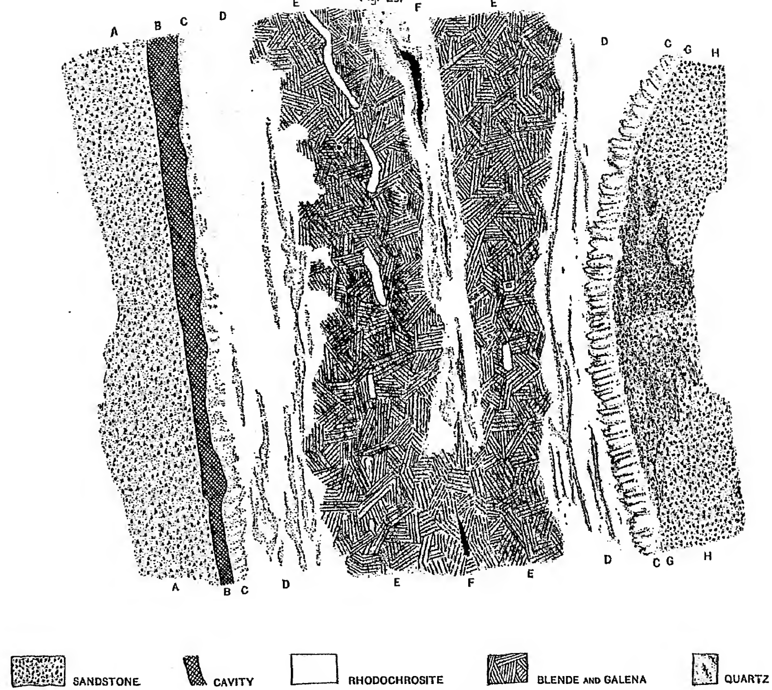
The vein is about a foot wide and is composed of a regular symmetrical arrangement of diverse minerals. The center of the ore is marked by a seam of quartz. The most remarkable feature of the section, however, is that while on the hanging the ore is frozen hard to the sandstone, on the foot there is an actual vacancy separating the ore from the country. This extends for a few feet above the place of the section, and is seen to find its downward termination as soon as the foot-wall penetrates into limestone, where the contact of the ore and the encasing rock is only marked by a slight selvage.

This description will render more intelligible the meaning of the detailed section of the vein presented in Fig. 25, which is intended to portray as accurately as possible the characteristics of the ore-occurrence at the point in the lode marked A on the outline-drawing, Fig. 26.

The main features are as follows: The western boundary of the vein is fairly straight, dipping, as the vein does, eastward. It is separated from the country-rock, a light-gray medium-grained sandstone, A A, by an actual vacant space, B B, of about half an inch. The edge of the ore nearest the foot-wall consists of an irregular band, C C, of about $\frac{3}{4}$ of an inch of quartz, speckled with pyrite and chalcopyrite. Toward the bot-

* According to Mr. C. W. Goodale, the manager.

Fig. 25.



RIBBON STRUCTURE

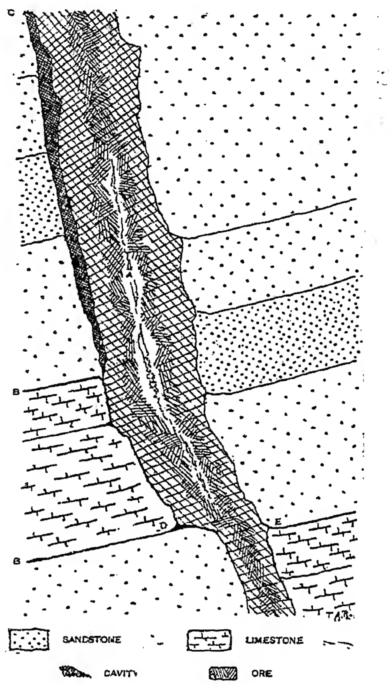
JUMBO VEIN, ENTERPRISE MINE, RICO, COLORADO.

SCALE 1 2 3 INCHES

tom this quartz narrows, but becomes clearly defined into a crystalline comb, with teeth at right angles to the vein.

Then comes a zone, D D, averaging $1\frac{1}{2}$ inches, of pink rhodochrosite. This band is broken into by veinlets of quartz, some of which are only branches from the outer seam, C C, while others traverse the rhodochrosite in bluish-gray streaks parallel

Fig. 26



to the general structure, and are peppered over with particles of pyrites.

The rhodochrosite band is broken on the right by the irregular outline of the blende and galena, E E, which is about 2 inches wide. There are blotches of yellowish "resin-blende" and patches of bluish-black galena distributed throughout a dark and intricate mixture of these minerals. They shade out into the white quartz, F F, which makes a bilaterally symmetrical division in the ore. The dark mass of the sulphides encloses shreds of rhodochrosite having distinct outlines.

Along the median line of the central quartz-seam, F F, occurs a succession of geodes, lined not only with crystals of the quartz itself but also with beautiful crystals of stephanite. The latter are seen in irregularly distributed clusters. In the outer quartz there are numerous specks of pyrite.

The eastern half of the vein presents, in reverse order, an exact repetition of the mineral bands just described. The separation between the rhodochrosite band and the outer quartz is more distinct. The dark sulphides also present a cleaner outline. The outer quartz has its comb-like structure strongly developed, the points of the crystals penetrating into the pink rhodochrosite and their base gradating into the dark-gray sandstone of the hanging-wall. There is not the slightest selvage or parting of any sort. The quartz is, as the miners say, "frozen" to the sandstone. The latter is marked by clouds of dark mineral hardly defined enough to be described as dendritic. This feature is traceable to the diffusion of minute particles of pyrite and stephanite. The rock is rich enough to be classed as ore.

In interpreting this structure, shall we follow the explanations given for the repeating symmetry of the comb-structure of the Drei Prinzen vein,* and accept the theory of successive crystalline growth from the sides of a gaping crevasse? Or are we to conclude that the mineral aggregates, now forming the ore, were derived by the substitution, bit by bit, of rock in place by material deposited from solutions circulating along the line of fissuring?

Do we conceive of veins as formed by the filling of pre-existing cavities, whatever their shape may be, produced by the rupturing of the earth's crust, or do we believe that lodes can be formed without any previously prepared vacant space and simply by the chemical interchange vaguely covered by the term metasomasis? or, again, do both these explanations find corroboration in the daily observations of the mine?

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* As drawn by Von Weissenbach in his book published at Leipzig in 1836. Other notable examples of this structure are C. Le Neve Foster's drawing of the Wheal Mary Ann lode (in the *Transactions* of the Royal Geological Society of Cornwall, vol. ix., 1875), and that of the Carn Marth lode by J. H. Collins (in the *Proceedings* of the Institute of Mechanical Engineers, 1873). Reference may also be permitted to the writer's four colored drawings of the Eureka, Songbird, Jumbo and Kitchen veins, accompanying a paper entitled "Vein Structure in the Enterprise Mine," in the *Proceedings* of the Colorado Scientific Society for 1895.

Walking recently along the railroad grade between Anaconda and Cripple Creek, in El Paso county, Colorado, I found

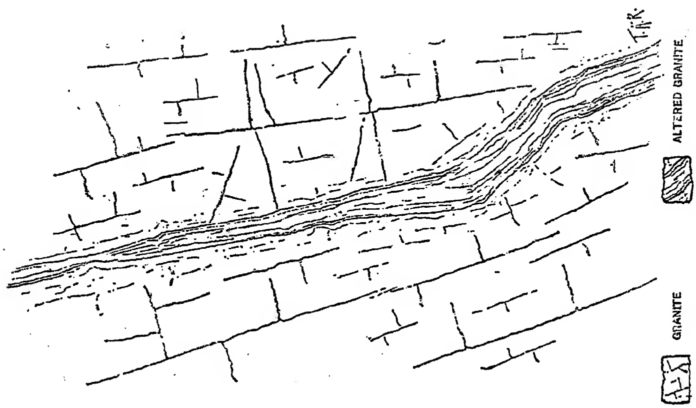


Fig. 28. A VEIN.

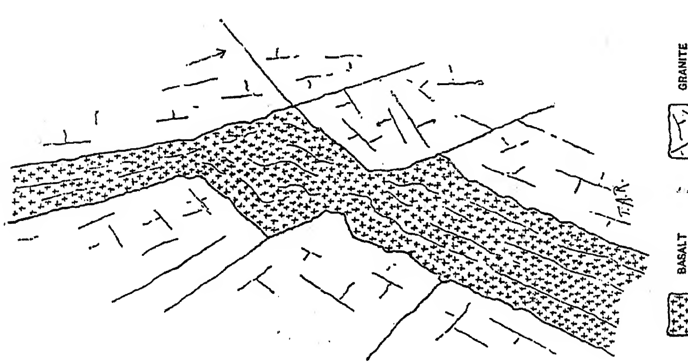


Fig. 27. A DIKE.

in the sides of two open cuts the testimony transcribed in Figs. 27 and 28, one representing a typical dike and the other a typi-

cal ore-vein. In both cases the country is the coarse-grained, red granite of the Pike's Peak region. In both the jointing is well developed. The dark dike of basalt in Fig. 27 cuts clean through the red granite. Its boundaries are clear, there is no mistaking the line of separation. Moreover, it is evident that the walls have duplicate outlines and that rupturing has separated them without the destruction of their definition. The throw of the fault-fissure followed by the dike can be seen to be about 14 inches, and its direction is indicated by the arrow.

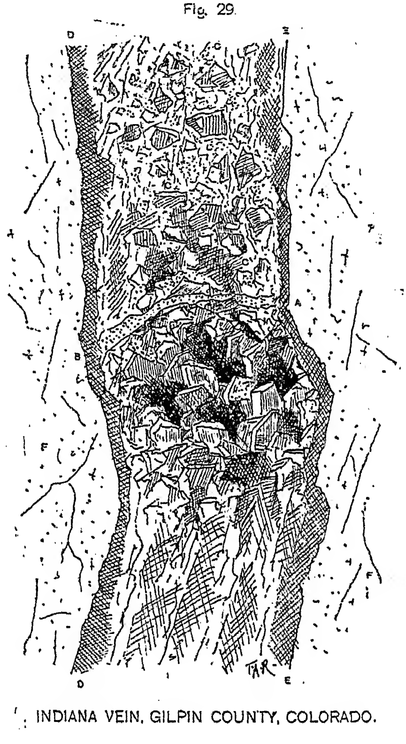
Fig. 28, sketched in the immediate neighborhood, illustrates a gold-bearing vein in the same granite formation. Here there is no essential difference between the country and the vein-filling. The latter is altered granite, easily recognizable as such, in spite of its having become granular and soft through the kaolinization of the feldspar. The walls of the vein are ill-defined, the streakiness of the filling being dimly repeated in the encasing rock. The vein-filling assays $2.60 gold per ton at this place, but is richer, without other material change of character, a few rods distant.

The dike, Fig. 27, is composed of foreign matter filling an evident fissure; the vein, Fig. 28, is rock in place changed into ore by the removal of some of its constituents and the substitution of new ones. In the former case liquid material rose into the fissure, probably *pari passu* with its formation. On the other hand, the vein of gold-ore traversing the granite gives no evidence of the occupation of a fissure by the incoming of new material. The ore is granite in place, softened, decomposed, discolored, and impregnated with gold, but still granite, clearly enough. Some liquid more subtle than molten lava was the vehicle which brought in the minute particles of gold and removed the alkali of the feldspar. It was water, circulating for long periods, and patiently searching out its way, which quietly changed the granite into gold-bearing ore.

Is it necessary in this case, as in that of the neighboring dike, to suppose the existence of an open fault-fissure? The evidence of a fault along the course of the vein cannot be discovered with certainty; nevertheless, judging from analogy and experience, we would certainly believe that the gold has been deposited along a line of displacement. It seems difficult to conceive that any fracturing, such as marks the beginnings of vein-forma-

tion, can take place without some displacement, however slight, of the two opposing rock-faces. Without such dislocation, though it be comparatively insignificant in amount, the fracture is only latent, and can hardly be said to exist. The possibility of a simple rupture, without any shearing movement or relative displacement, cannot be denied;* but observation underground indicates that, so far as the deposition of ore is concerned, we

Fig. 29.



INDIANA VEIN, GILPIN COUNTY, COLORADO.

have invariably to deal with rupturing accompanied by a relative displacement of the rock-walls. In other words, veins are generally built on fault-lines. The absence of evidence of such movement in a section on one particular plane is not conclusive, since the displacement may have been at right-angles to the section.

* In this connection I would refer the reader to the suggestive paper of Mr. William Glenn on "The Form of Fissure-Walls, as Affected by Sub-fissuring, and by the Flow of Rocks," read at the Atlanta Meeting of the Institute, October, 1895, and printed in *Trans.*, xxv., 499.

Where a vein cuts across sedimentary rocks, the dislocation may be looked for along the bedding-planes. Such is the case at Rico, in the Enterprise mine already referred to, where the breast of a stope will show a vein traversed by a fracture at right-angles to its walls, and apparently unaccompanied by any dislocation, but further examination will frequently show that there has been a displacement of the country along the dip of the sandstone and limestone beds, in the strike of the vein itself.

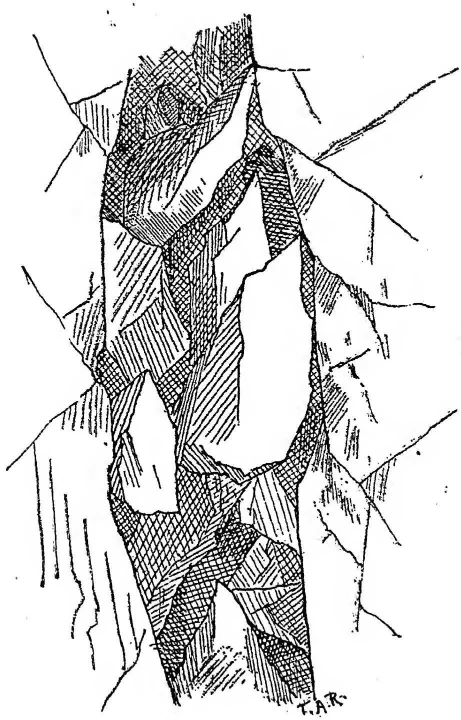
The question here arises, whether the formation of the ore-vein required the existence of an open fissure. In the particular case shown in Fig. 28, the quantity of foreign material within the vein is insignificant in amount; the "ore" being simply altered rock in place. That this rock became mineralized by the penetration of metal-bearing waters was probably due to the crushing of the granite by an original slight faulting movement, presenting facilities for circulation and consequent chemical interchanges. Minute spaces there probably were; but a clear opening, or a slow crevassing, such as accompanied the formation of the neighboring dike, seems hardly needed. The ribbon-structure of the Enterprise section, in Fig. 25, presents features much more difficult to explain.

When Werner and his school attributed the filling of veins to the agency of descending waters, the existence of open fissures at the time of vein-formation was conceivable, because the theory necessarily restricted such operations to the vicinity of the surface. But the acceptance of ascending waters as the main agents of ore-deposition, and the recognition of the conditions possible to the formation of large masses of sulphides, at once transferred the laboratory of ore-formation to a deeper horizon; and the suggestion that veins were filled by the deposition of layers of mineral precipitated from waters passing upward along fissures which were kept wide open during such time as was required for crystalline growth to choke them with ore, was immediately ridiculed by the miner, because his daily experience taught him that the vein once deprived of its filling did not remain open, but was inevitably closed by the pressure of the surrounding rock. In many cases, in the absence of artificial means of support, his mine-workings collapsed, so that where there was once a level wide enough for a man to walk

through, there came to be only a seam of mud enclosed in shattered rock.

Despite the miner's objection, however, there is evidence that fissures do sometimes occur, which have been sufficiently open to permit the tumbling in of large pieces of rock. Such

Fig. 30.



MAMMOTH MINE, ARIZONA.

an occurrence was observed in connection with certain faults which disturb the Virginie lode at Roure, near Pontgibaud, in France,* where, at a depth of 164 feet from the surface, a fault-fissure encloses a mass of clayey material containing boulders of a black, soft, and porous rock, which can be identified as

* See *Étude sur les gîtes métallifères de Pontgibaud* par M. Lodin, Ingénieur-en-chef des Mines. *Annales des Mines*, April, 1892, and "The Lodes of Pontgibaud," by the writer, in *Eng. and Min. Jour.*, August 11 and 18, 1894.

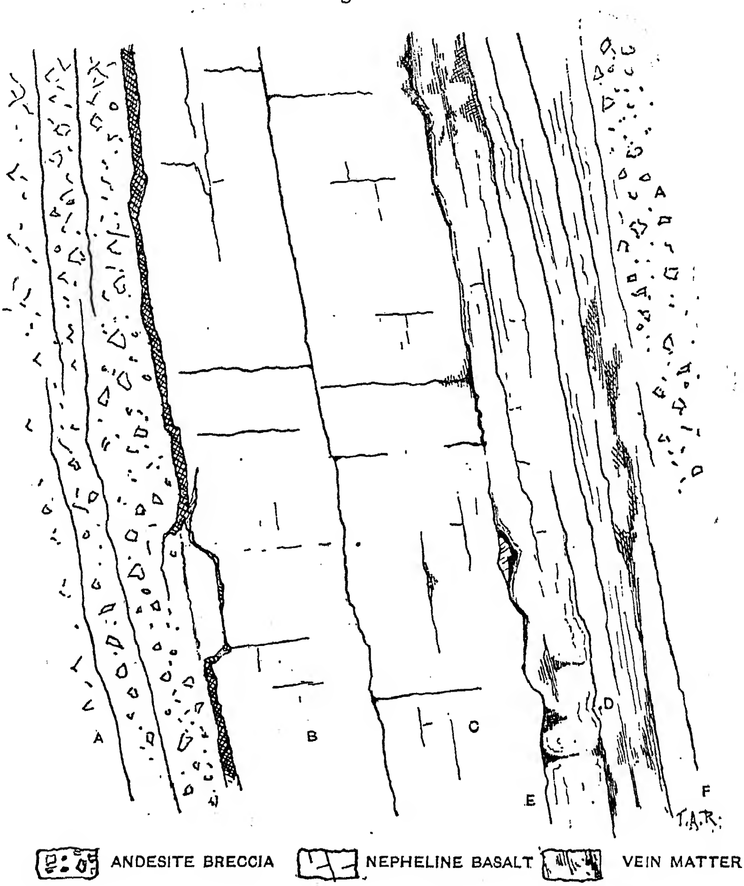
pieces of scoriaceous lava. No such rock occurred elsewhere underground; and the boulders must have been portions of the Quaternary alluvium which covered the outcrop of the lode, and fell into it at the time of its intersection by an open fissure, which long post-dated the formation of the ore-vein itself. The mines are in a district which has frequently been subjected to earthquakes, and in the heart of a region formerly the scene of great volcanic activity.

We must be careful, however, to distinguish between the formation of cavities within the zone of the vadose circulation, and their existence in "the deep," where sulphide ores have their origin.

Two examples may be quoted. The first is shown in Fig. 29, sketched November 25, 1895, in the stopes above the 800-foot level in the Indiana mine, Gilpin county, Colo. The lode, which is the California vein, in its extension westward from the Hidden Treasure mine, is about 2 feet wide. There is no parting or selvage separating it from the country. The latter is a quartz-feldspar rock, best described as granulite. Near the lode it is seamed and sprinkled with pyrite, and sufficiently gold-bearing to be sent to the stamp-mill. The main pay-streak is almost entirely composed of black zinc-blende which, by candle-light underground, contrasts strongly with its encasement of light-gray country. The upper part of the vein, in this particular stope, consists of a breccia of zinc-blende, with an occasional spattering of wall-rock, the latter so disintegrated as to resemble gravel. At one point, A B, there is a shred of wall-rock lying across the vein. Lower down there are a number of cavities or vugs scattered among angular fragments of ore. It all looks loose, like an old stope filled with ore that has been mined, but the material is hard and difficult to detach without explosives. Lower again, the vein loses both its cavernous and its brecciated character, and consists of a compact body of blende. It may be added that, even where the brecciation is most evident, both walls are lined with a few inches of ore unbroken and firmly attached to the wall-rock into which it gradates. The vugs, when first found, were full of gas (CO_2, probably) and the miners suffered from bad air when working in ground of this character. The pieces of blende are held together by a siliceous cement, which also covers each fragment in the form of a gray-

blue chalcedonic coating. It is almost certain that the cavities above described contained water, previous to the drainage of the ground by the penetration of the level underneath.

Fig. 31



THE MOOSE VEIN, CRIPPLE CREEK, COLORADO.

Another instance is suggestive. In the Mammoth mine, Pinal county, Ariz., already described, the granite in the east cross-cut at the 300-foot level, north, has an extraordinary number of fissures partially occupied by broken pieces of rock, so wedged in as to leave open spaces. The pieces are not of any foreign rock, but are identical with the enclosing granite. Fig.

30 is a reproduction from a sketch made on the spot, March 15, 1893. The elongated cavities, such as that illustrated, were found full of water when first reached by the cross-cut; but they became drained as the workings tapped them, and thereby depressed the water-level of the mine.

This instance suggests why mining excavations collapse, and yet a natural cavity underground might remain open. The former contains unconfined air only, while the latter may be filled with a confined and practically incompressible fluid, water.

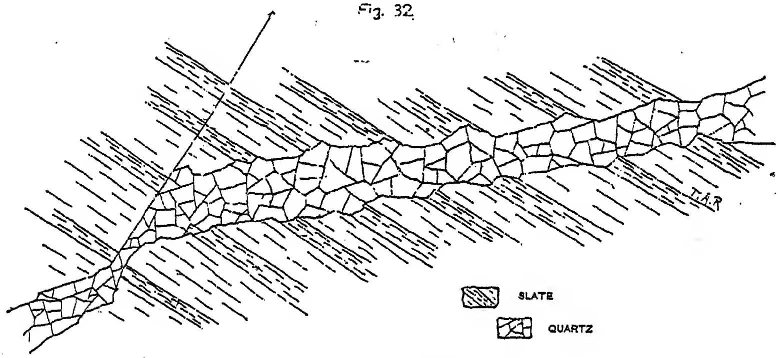
It is the usual experience in mining that when the abandoned workings of a mine are flooded they are less likely to collapse than when they are dry. This is due partly to the exclusion of air, and partly to the sustaining power of the water itself, as suggested by Mr. P. Argall, in the *Eng. and Min. Jour.*, September 23, 1893, p. 314.

The formation of the hollow spaces occasionally seen in veins is, I believe, in most cases subsequent to the ore deposition, and may therefore have taken place at a time when erosion had brought that portion of the vein near to the surface. The Indiana section, Fig. 29, shows that the cavities have been produced by the shattering of a vein of zinc-blende already formed. The only occurrence of later date is the consolidation of the mass by the agency of water bearing silica, unaccompanied, so far as can be seen, by the deposition of any metallic minerals. In the Mammoth mine, Fig. 30, the blocks of rock wedged within the cavities were coated with crystals of vanadinite and wulfenite; but there seems to be no connection between the presence of these later minerals and the formation of the ore-bearing parts of the lode. They are the result of secondary processes, of which the upper part of a lode is the characteristic zone of activity.

The vein in the railway-cut, cited above as a type, presents a filling readily recognizable as simply altered rock containing only an insignificant percentage of material foreign to the composition of the original granite. Nor is this an abnormal type of vein-structure. The rich gold mines on the adjacent hills afford numerous examples of this very kind of lode-formation. (And incidentally I would say that I know of no mining district which illustrates modern views on ore-deposition so clearly

as does Cripple Creek.) Of such mines I would quote the Independence vein, whose richness is such as to cause its commercial value to obscure its scientific interest. It does illustrate very aptly, however, this part of our enquiry, because the ore is so very evidently only altered country-rock. In 1893, when the workings had not penetrated far from the surface, the car-loads of ore sent from this mine to the Denver smelters gave the impression that some one had blundered and either shipped waste from a cross-cut or else switched cars of ballast into the place of loads of ore. One could see that it was the normal Pike's Peak granite with its big pink feldspar, but

Fig. 32



DRUMLUMMON MINE. MONTANA.

it required a trained eye to note that the mica had been largely removed, leaving small iron-stained patches. It was ore by courtesy, because there was enough gold present to give it a certain commercial value; but to the petrographer it was clearly granite, not much altered and but slightly mineralized.

The vein leaves the granite and, going northward, penetrates into andesite breccia. Its character remains the same; the ore is still altered country-rock; only now it exactly reproduces the structure of its new encasement, and the habit of the andesite breccia is quite evident, although blotches of sylvanite and fluorite may occasionally try to obscure it. The strike of the vein, its width, its richness, all appear unaffected by the passage from one formation into the other, while the change in the structure of the ore is so marked as to render it easy for the observer to know what is the enclosing rock without looking at the walls.

In a case such as this—and it is not abnormal—it is not necessary to suppose the original existence of an open cavernous fissure, since the material of the vein is the material of the rock which was there before vein formation began. The vein follows a line which became a path for metal-bearing waters. Minute interspaces there probably existed, such as would be produced by the crushing and slight dislocation of particles of rock lying along a line of fracture; but a clear opening, a crevassing, such as accompanied the origination of the dike, seems hardly needed.

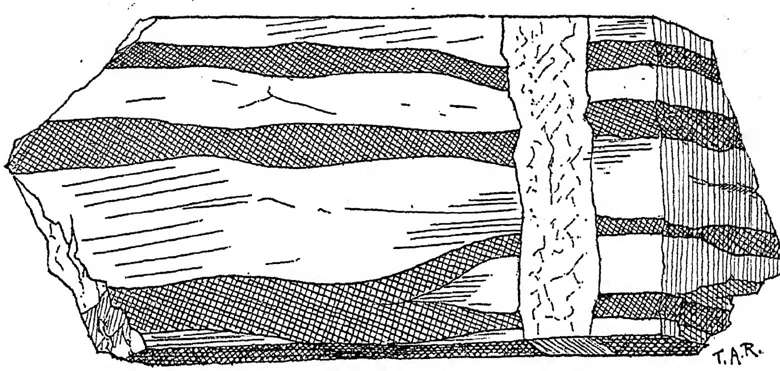
Occasionally, it is true, we do find veins full of minerals foreign to the encasing rock and so symmetrically arranged in bands having a comb structure as to suggest to many investigators that they were formed by successive crystalline growth from the walls of a vacant fissure. Such, no doubt, would be the interpretation given to the section of vein illustrated in Fig. 25. The reversed repetition of the quartz, rhodochrosite and sulphides is evident enough; but the most striking feature to me is the equal width of each of the two bands of the same mineral. Each vein of mineral would seem to have been fractured exactly in the middle previous to the deposition of the next succeeding one.

This specimen, and numerous similar structures in the same mine, indicate that the rhodochrosite was the first laid down, replacing, in part at least, the crushed rock which encased an original line of fault-fissuring. Subsequently another fracturing occurred, and this time the line of least resistance was the rhodochrosite itself, which, being homogeneous, broke down its center. The shattered carbonate offered an easy prey to the sulphide-bearing waters which laid down the blende and galena. The presence of bits of rhodochrosite within the sulphide band indicates that the substitution was irregular. Later, new disturbing forces were at play and the vein was fractured not only along its middle, as heretofore, but also along the lines of its contact with the encasing rock. These fractures were healed by the deposition of quartz, accompanied first by iron and copper pyrite, and then by rich silver-bearing minerals, such as the stephanite. The corrosion of the sandstone on the hanging had on that side irregularly widened the vein so as to give it greater strength; therefore the next movement, the last, took place

along the foot-wall. This apparently resulted in nothing save the crushing of some of the encasing rock and the formation of a selvage whose removal produced the cavity which was so striking a feature of the stope.

Another typical illustration of this structure is presented by the Amethyst-Last Chance vein (at Creede, Colo.) which is certainly a magnificent example of an ore-break.* The country-rock, trachyte, has undergone multiple fracturing and ore has

Fig. 33.



NATURAL SIZE

DRUMLUMMON MINE, MONTANA.

been deposited along the division-planes so that there are walls *ad libitum.* The regular ribbon-structure produced by the deposition of agatized quartz in a sheeted rock is very beautifully marked, and the same process of silicification is further evidenced in those places where the lode consists of breccia composed of pieces of country covered by concentric layers of agate. The lode itself is much wider than the pay-streak of silver-ore, which usually follows the foot-wall. On the hanging the boundary between vein and country is fairly discernible; on the foot less so, because for several feet beyond the ore there is a red jasperoid which gradates into country.

* At Red Mountain, in Ouray county, Colo., it has been the practice to speak of the veins (the Guston, Yankee Girl, and other celebrated lodes) as "ore-breaks," a break in the rock accompanied by ore—a term, it seems to me, much preferable to "fissure vein."

In the Enterprise example, Figs. 25 and 26, each succeeding fracture occurred in the mineral deposit which had healed the previous fracture. In other instances the mineral deposit appears to have proved harder than the encasing rock and the second fracturing took place near the original one, but in the soft rock rather than in the hard vein, thereby producing a new break parallel to the first one, and close to it, causing a repetition of vein-walls such as have already been described in connection with the sections given in Figs. 1, 9, 10, 15 and 21. Or there may be the production of companion-fissures forming contemporaneous veins, such as are shown in Figs. 13 and 19. Finally, the companion-fissures may be so multiplied as to cause a sheeting of the country and the formation either of one vein and several, subordinate, smaller and parallel to it, as in No. 15, or of a series of ore-streaks united by mineralized country so as to form one large lode, as seen in Figs. 5, 14, 17 and 31.

The evidence of a multiplicity of fracturing, whether successive or contemporaneous, is the clue, I venture to believe, to many of the anomalies of vein-structure. No district within my knowledge so well illustrates this aspect of the inquiry as Colorado's new El Dorado, Cripple Creek, in El Paso county, where gold-veins occur as mineralized and enriched portions of dikes, phonolite and basalt, traversing masses of andesite tuff and breccia. Other types are observable, but these are to-day the most characteristic. The mineralized rock forming the vein and that less distinctly gold-bearing country which encloses it, have been subjected to such multiple fissuring as to produce a very marked division of the rock into parallel bands or sheets which may be a fraction of an inch apart or several yards. This structure can be seen no less in hand-specimens than in blocks an acre big. The Moose vein, on Raven hill, is a fair example. It is illustrated in Fig. 31, as seen October 27, 1895, in the back of the sixth (or 350-foot) level. A is andesite tuff and breccia, B C D is a dike of dark, blue-gray nepheline basalt, subdivided into two barren parts, B and C, and one ore-bearing portion, D. Native gold and telluride compounds (sylvanite and calaverite) occur along the seams in the basalt where it is decomposed and iron-stained. The pay-streak extends from E to F, about 10 inches.

This sheeting or multiple fissuring was probably the result of

shrinkage accompanying the cooling of the volcanic rock. The fractures have a contemporaneity of origin quite distinct from the successive ruptures discussed in connection with the ribbon-structure of the Enterprise section. The latter were marked by the precipitation of diverse minerals, while those of a Cripple Creek vein are characterized by a similarity of mineral deposition.

Cases also occur where there can be discerned a combination of both these types of multiple fissuring.

A line of weakness, or even a region of weakness, once developed in the earth's crust is apt to continue to be a line of least resistance available for future fracturing. Even when a quartz-vein is formed along a line of rupture, healing the break and strengthening it with a substance harder than the rock-walls themselves, we may suppose that the next break will take place along the line of weakness presented by the imperfect cohesion existing along the plane of contact between the hard quartz and the less resisting rock.

The gradual penetration of mineral solutions into the immediately encasing country may finally obliberate the divisions due to multiple fissuring. The sheets of rock separating one from the other would be replaced by ore, and nothing might remain of the original structure save faint partings in the lode, such as are less evident to the eye than to the hand of the miner who instinctively uses them to assist him in breaking ore.

Thus, I believe, the collection of observations in various mining districts tends to the modification of that idea of clean-cut definition which accompanied the early ideas of vein-structure. The evident contact between two dissimilar rocks, such as is seen along the walls of a dike, will be often found in veins to be replaced by an indistinct gradation from mineralized to unmineralized rock, originally the same but now rendered unlike by the selecting action of chemical solutions.

We are justified, however, in putting some limit to the depth of possible ore-formation, since that formation is dependent on the presence of water. The record of the largest number of careful observations has shown that as we sink into the earth the increment of temperature is 1° F. per each 48 feet of descent. At this rate the critical point of water would be reached at 34,704 feet or 6½ miles from the surface. Where the tem-

perature is that of the critical point (773° F.) water cannot exist as a liquid no matter how great the pressure, but becomes dissociated into its gaseous elements. Moreover we are warranted in believing that the thermometrical gradient becomes more rapid at depths beyond those reached by human observation because of a decreased conductivity in the rocks, or as Professor Prestwich, the best authority on these matters, puts it:*

"Taking into consideration the probable limitation of the percolation of water, and the possible diminution of conductivity with increase of depth, if there should be any alteration in the thermometric gradient, at great depth, it will be more likely to be in the direction influenced by these more or less certain factors."

Therefore, taking these conditions into consideration, we may expect the circulation of water to cease at 20,000 feet or thereabout. But at the maximum depth the maxima of temperature and pressure must obtain. It must necessarily be a horizon of solution. Precipitation would hardly begin until a lowering of the temperature and a lessening of pressure permitted it. The deposition of ore is the direct result of precipitation, therefore actual ore-formation is likely to be limited to a depth often of 15,000 feet.

It is not difficult to surmise why clean-cut fractures are not necessarily most favorable to ore occurrence. In the Drummummon mine, Montana, the distribution of the ore appears to be connected with the change in the angle of intersection between the course of the veins and the strike of the slate country. Most of the ore-bodies have been found where the course of the veins (N. 15° E.) cuts the slates at an oblique angle and the levels run out of ore when their direction is either at right angles to, or conforms with, the strike of the country (N. 17° W.).

Fig. 32 is a sketch made in one of the surface-workings of that mine which illustrates in miniature the fact above noted. It represents a small quartz-seam 2 inches wide, traversing the slates, whose structure is very clearly marked by the color bands following lines of original sedimentation. Near the left of the sketch the quartz follows a joint and becomes narrowed, while where it crosses (along a line of slight dislocation) the country it has irregularities and bulges which answer to the alternating slate-bands. A rough, ragged fracture, when continuous, may

* *Controverted Questions of Geology*, by Joseph Prestwich, D.C.L., F.R.S., etc., Macmillan, 1895, p. 247.

be expected to offer more surface to solvent action and more, but not too many, obstacles to a rapid circulation of underground waters. Its structure also means more opposition to the closing in of the walls, because the irregular faces of the fracture, when they come together, leave openings which, if not along one section then along another, have intercommunication, and so permit of a passage which would be badly impeded, if not absolutely stopped, by the closing in of smooth walls.

Fig. 33 represents, to actual scale, a piece of slate enclosing a quartz-vein, which came from near the end of the 700-foot level, also in the Drumlummon mine. It so happens that this is a true illustration in miniature of what the lode itself is doing at this point. The New Castletown lode, on which the level is driven, is at this point cutting at right-angles across the bedding of the slates and is barren of ore. In the hand specimen, reproduced in the drawing, a quartz-vein, not quite half an inch wide, cuts perpendicularly across the slate whose bedding is rendered beautifully marked by dark bands. The vein has a uniform width, it has regular well-defined walls guiltless of the projections and bulges noticed in the previous illustration. It may be only a convenient coincidence, but it is a fact that the quartz in Fig. 25 was opalescent and destitute of other minerals while that in Fig. 24 was true ferruginous vein-quartz.

Thus underground work bears daily testimony to the close dependence of ore-occurrence upon the geological structure of the enclosing country, a relation, the importance of which Mr. S. F. Emmons has done invaluable service by clearly stating in more than one of his contributions to the *Transactions*. Wanting a proper understanding of the structure of the rock encasing his vein, the miner gropes but blindly in a maze of tangled phenomena until the geologist, by their proper elucidation, gives him a light which dissipates much of the darkness obscuring his progress underground.

The Sulphuric Acid Process of Treating Lixiviation Sulphides.

BY FREDERIC P. DEWEY, WASHINGTON, D. C.

(Colorado Meeting, September, 1896.)

THE improvement in leaching introduced by the Russell process has stimulated the development of processes for refining lixiviation-sulphides.

In the early days several processes for dealing with the sulphides were proposed, and some of them were tried more or less; but the business finally settled down to sending the sulphides to the smelters, although this was known to be both troublesome and expensive. In 1891 Mr. C. A. Stetefeldt* introduced at the Marsac mill, Park City, Utah, an unpatented process, built up out of the general fund of information available. It consisted in matting the sulphides, grinding, roasting, grinding again, and dissolving the copper out in diluted sulphuric acid, then melting the silver and crystallizing the bluestone. It did not yield fine bullion, hence the bullion had to be refined as well as parted; besides, there was some loss. This process was thoroughly tried at the Marsac refinery and then a year's run was made, the net result of which was that it did not prove sufficiently better than the sending of the sulphides to the smelters to warrant its substitution for that practice.

In 1893 the Dewey-Walter Refining Company undertook the refining of the Daly sulphides in the Marsac refinery by the sulphuric acid process, upon which two United States patents† have been issued to the writer. Naturally, difficulties were encountered in starting a new process, and most of 1893 was taken up in getting it into smooth working order; but in 1894 a run was started, in which all the regular sulphides produced by the

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* *Trans.*, xx., 37; xxi., 286; xxiv., 221; "Lixiviation of Silver-Ores with Hyposulphite Solutions," C. A. Stetefeldt, 2d ed., 1895, p. 158.

† Nos. 490,063 and 561,571.

Marsac mill in that year were refined, and thus complete statistics of operation by this method were obtained.

Broadly speaking, the process consists of six main operations:

1. Boiling the sulphides with strong sulphuric acid in an iron pot;
2. Dissolving out the sulphate of copper and silver in a lead-lined tank, leaving a residue containing the gold and lead of the sulphides, and also rich in silver;
3. Precipitating the silver out of the filtered solution by copper plates;
4. Sweetening,* drying, pressing, and melting the cement-silver;
5. Treatment of the solutions after the removal of the silver to crystallize the sulphate of copper, and recover the excess of acid for re-use;
6. Treatment of the gold-bearing residues.

The 1894 run of the Marsac leacher produced 116,519.5 pounds of regular sulphides which were treated by this process. For convenience they were divided into 25 lots, mostly from 4500 to 5500 pounds in weight. As reported by the assayer of the Daly Mining Company, these lots varied in composition as follows:

Composition of Daly Sulphides for 1894.

	Ag. Oz. per ton.	Au. Oz. per ton.	Cu. Per cent.	Pb. Per cent.
Maximum,	11,127	14.8	32.9	0.2
Minimum,	7,835	7.6	20.3	0.6
Average,	9,827.4	11.225	27.17	0.33

The totals were:

	Ounces.
Silver,	572,544.4
Gold,	646.1
	Pounds.
Copper,	31,585.3
Lead,	385.6

These determinations of lead are too low, the actual average for the year being about 2.5 per cent.

I have found the percentage of free sulphur in the Daly sulphides to vary considerably. It is generally rather low, and

* "Sweetening" is washing. "Sweetened" silver is simply well-washed silver.

sometimes very low. Lots 8, 9, 10, 11 and 12 showed 3.3, 3, 2.72, 0.48 and 1.24 per cent. of free sulphur respectively. The total sulphur was found on one occasion to be 20.74 per cent.

Aside from the soluble salts remaining in the sulphides through imperfect washing in the filter-press, the drying causes oxidation, and soluble sulphates are produced. As received by the refinery, the sulphides contain several per cent. of soluble salts and small amounts of iron, lime, antimony, arsenic and other impurities.

Plant.

The plant required is simple and well known, and easily managed without specially skilled labor. It consists of two ordinary cast-iron pots, such as are used in parting bullion; a series of 21 lead-lined tanks for dissolving, filtering the solutions, precipitating the silver and filtering off and sweetening the cement-silver, together with crystallizers to recover the blue-stone and evaporators to concentrate the mother-liquors for re-use in the pot; a dryer and press for the cement-silver; a furnace for melting the bullion; 4 storage-tanks for acids, pumps for handling the liquids, and a pulverizer.

Boiling-Pots.—These are ordinary cast-iron pots. The large one for boiling sulphides is 47.5 inches in diameter and 3 feet deep inside, with a rim 3.5 inches wide, slanting slightly to the inside. It is 1 inch thick on the bottom and ½-inch on the sides. Each pot weighs about 1500 pounds. It rests on a cast-iron plate 5 feet 7¼ inches by 5 feet 6½ inches by ½-inch, with a 2-inch rim to fit over the brick setting. At first we obtained our pots as special castings from a New Jersey foundry, paying a corresponding price for them; but afterwards we obtained them at a smaller price from the foundry of Davis, Howe & Co., at Salt Lake City, who, moreover, take the old pots from us as scrap-iron at a good price. Their pots have done even better service than the New Jersey ones. During 1894 the pots lasted from five to six weeks, but during 1895 we have had better results. From March 1, 1895, to March 18, 1896, 7 pots were used, with an average life of seven weeks and five days, and treating an average product of 23,737 pounds each. The acid does not dissolve much iron during the boiling, but a net-work of cracks develops until the acid leaks through. With proper

care there is little danger of loss of silver on account of the pastiness of the charge. About a day is usually lost in changing pots. Some of the pots were made 2 inches thick on the bottom; but they were no better than the 1-inch pots. The average loss in weight per pot was 67 lbs., as shown by the difference in weight of 7 of the 94 pots when new and when sold as scrap-iron.

The pot is set in brickwork over an ordinary grate, in which the local coal from Coalville, Utah, is burnt.

The top of the pot stands 14 feet from the ground-floor, so as to allow all the solutions to travel downwards from tank to tank by gravity. Over the top of the pot a spherical cast-iron hood, 49 inches in diameter and ½-inch thick, is placed, having a working door 18 inches by 9 inches on the side. At the top is an opening with a water-joint cast in, into which fits a 10-inch lead pipe, 6 feet long. This leads to a lead-lined stack going through the roof. This arrangement is to remove the fumes given off during the boiling. The draft can be increased by a steam-jet in the stack, as may be occasionally necessary.

The small pot for re-treating the residue is 2 feet 8 inches in diameter and 2 feet deep, resting on a plate 4 feet 3 inches square. This pot has a conical hood, made of sheet-lead over a frame-work of iron covered with lead. The large pot originally had a lead hood, but this was replaced by the cast-iron one, which has given better satisfaction. The foundation for the large pot is built up solid from the floor, but for the small pot 12-inch walls are run up to the fire-box level, leaving a chute for the discharge of the ashes.

The tools required at the pot are a hoe for stirring, a discharging-ladle, coal-scoop, short-handled shovel, broom, lead-bucket and a trough to carry the boiled charge to the dissolving-tank. This latter is a light V-shaped trough of sheet-iron, which is placed in front of the pot when ready to discharge, and leads to the dissolving-tank.

The Dissolving-Tank.—This is 4 by 8 by 2 feet, made of 2-inch rough boards dowelled, with the ends let into the sides ½-inch and bolted at the ends through buck-staves. All the tanks are made in the same way and thoroughly painted with asphalt. The first dissolving-tank was lined with 8-pound lead; but this was not heavy enough, and 12-pound lead is now used.

The contents of this tank must be heated by the introduction of steam. If this is done directly from a pipe attached to the lining, the vibrations set up will soon open a joint in the lead lining and cause a leak. To avoid this, a very simple arrangement was formerly used. It was a lead cone 15 inches long, fastened to the steam-pipe. It was perforated with $\frac{3}{16}$-inch holes on the top and sides, and stood on feet about 2 inches from the bottom of the tank. The steam-pressure was gradually reduced by the cone and the solution heated up gradually and quietly. This arrangement worked well, but the cone soon wore out. At present a plain lead pipe, turned up at the end and entirely free from the lining, is used with satisfaction.

This tank is provided with two discharge-hose, one for each filter-tank, arranged to draw off the solution as free as possible from fine residue. Pieces of lead pipe 10 inches long, projecting through the end of the tank, are burned to the lead lining at the bottom of the tank. A piece of rubber hose about 2 feet long is slipped over the inside end of the pipe. When drawing off the solution, the open end of the hose is adjusted, by a lead string, just below the surface of the liquid, so as to draw off only the clearest portion. This arrangement is a great saving for the filters, which at best are soon enough choked up by the fine residue. This tank has a $\wedge$-shaped lead-lined, counterpoised cover, with a triangular opening in the side for the introduction of the trough used to convey the charge from the boiling-pot to the tank.

Filters.—The two filters are each 3 feet by 6 feet by 2 feet, lined with 8-pound lead. Considerable difficulty was experienced at first in constructing a suitable filter. The difficulty was to get a filter that would retain all the residue, giving a clear filtrate, and at the same time give a good rate of filtration. If the residue passed through, it reduced the fineness of the bullion and at the same time very small amounts of gold appeared in the fine silver and were practically lost. A satisfactory sand-filter which, by washing from below, could be easily freed from fine residue after becoming clogged, was finally constructed as follows: Sheet-lead was first bent so as to make slats 2 inches high by 1.5 inches wide. The bottom was open and the sides were notched at the bottom to allow free passage for the liquids. The slats are put about 2 inches apart

on the bottom of the tank, and are 2 inches shorter than the width of the tank, leaving an inch at each end of the slats. On the slats is placed a plate of 8-pound lead perforated with $\frac{3}{16}$-inch holes $\frac{1}{2}$-inch from center to center, and then a thickness of cocoa matting. To make a tight joint between the slats and the side of the tank, a 6-inch strip of asbestos cloth is laid around and driven tightly against the sides by a tightly-fitting strip of hard wood. The asbestos laps over the cocoa matting about 2 inches. Next comes a second layer of cocoa matting and finally a second lead plate over all. On this foundation from 3 to 4 inches of clean quartz sand is spread and well-hammered down. A discharge-pipe runs out of the bottom of the filter at one end.

The hose from the dissolving-tank discharges into a perforated lead box on top of the sand, to prevent the stream from disturbing the sand. A steam-pipe for keeping the solution hot also discharges into this box. When new, this filter gives a large stream of clear filtrate. It must be carefully watched at all times. Its success may be shown by the fact, that notwithstanding the very fine nature of the residue and its liability to run through, 401 bars of bullion showed 999.5 and 45 bars 999, with an average of 999.4 fineness for the whole year.

The fine residue gradually chokes up the filter and in about 2 weeks the rate becomes so slow that it must be washed. This is done by running in water from the bottom and thoroughly stirring up the sand. The muddy water holding the residue in suspension is then pumped out into a small filter. This is repeated once or twice and the filter is smoothed off and hammered down, when it is ready for use again and will give a good rate of filtration.

Precipitating-Tanks.—The small precipitating-tank is 8 by 5 by 3 feet, lined with 8-pound lead. It is used for the first charge from the dissolving-tank, which contains more copper and is known as "copper-solution." The large precipitating-tank is 9 by 7 by 3 feet, lined with 8-pound lead and with an extra sheet of lead laid on the bottom. It was made large and of these dimensions to fit a space in the old refinery. It is used for the silver-solution. Each precipitating-tank is provided with an old syphon-pump for stirring the solution. A lead pipe reaches from the pump to the bottom of the tank, and when

steam is turned on, it is allowed to suck in air, which mixes with the steam and stirs the solution.

Silver Filters.—These tanks are 6 by 3 by 2 feet lined with lead and fitted with heavy asbestos cloth filters between lead plates, the whole resting on lead slats. One has an $\frac{1}{8}$-inch iron cover in three sections and is used to sweeten the silver in, and the other is a general filter-tank through which solutions pass, after precipitation, to catch any fine silver in suspension. Each filter has a sump consisting of a tank of the same size lined with 8-pound lead.

The sheet-iron cover is convenient and gives good satisfaction, but it has been the occasion of an annoying incident. The bullion fell off from 999.5 to 999; and after a thorough investigation it was found that some unusually large charges of cement silver had been put into the filter, and probably the sag of the cover in the middle had touched the silver or the wash-water and reduced a little copper from the wash-water. At any rate the fineness rose again after stopping the practice of washing such large charges.

Evaporators.—These are 8 by 4 by 2 feet lined with 8-pound lead. They have gridiron coils of $\frac{3}{4}$-inch lead steam-pipes connected together. Just enough steam is used to blow out the condensed water. They are housed in, the housing being provided with working-doors and a stack. They will evaporate away 75 per cent. of a 20° B. solution in 24 hours. A hose and launder deliver the concentrated solution to the crystallizers.

Crystallizers.—These 7 tanks are 3 by 6 by 2 feet, lined with 6-pound lead and provided with false bottoms. Each one has a draining-board and 12 2- by 2-inch wooden strips across the top, from each of which hang 6 lead strips 3 inches wide. The finest crystals separate on the strips. The blue-stone crystals on the bottom are chopped out with iron bars, the false bottoms being employed to save the bottom when this chopping-out is done.

Residue-Filters.—Two small tanks are used for these; one for the residue from the dissolving-tank, and one for the muddy water from washing the large filters. They have no asbestos and only 1 inch of sand, but they have a very slow rate of filtering. A layer of residue on the bottom soon chokes them up so that it is almost impossible to get anything through; and most of

the solution is usually syphoned off from the top after the residue has settled. Both filters discharge into the dissolving tank.

Acid-Storage.—The large storage-tank for 66° B. acid is 8 feet in diameter and 8 feet high, lined with 10-pound lead. It holds about 46,000 pounds of 66° sulphuric acid. The small tank is the old Roessler converter 4 feet in diameter and 6 feet high, and holds about 8000 pounds. Both these tanks have 2 stoneware cocks, one for every-day use, and the other a safety on the end of a length of lead pipe, ordinarily hung up at the top of the tank.

A section of 15-inch wrought-iron pump-column, holding about 1500 pounds of acid, is used as a *monte jus*, to elevate the acid by air-pressure, and two sections are used to store a supply of acid above the level of the pots for feeding them.

The evaporated acid is stored in a tank 5 feet 6 inches by 3 feet by 2 feet below, and in two round tanks, 3 feet 6 inches in diameter and 5 feet high, above, convenient to the boiling-pot.

Silver-Dryer.—After sweetening, the cement-silver is removed from the filter and put into sheet-iron drying-pans 2 feet 6 inches by 1 foot 2 inches by 3¼ inches, made of No. 10 iron. The bottom and sides are in one piece and the ends are riveted in. The dryer consists of a frame-work of angle-irons enclosed in brick-work. It holds 6 tiers of pans, 4 in each tier, or 24 pans in all. Under each one of the bottom 3 tiers of pans are arranged 5 4-inch steam-pipes joined together at the ends by cast-iron heads. When the dryer is full the pans are arranged so that the hot air zigzags below and above each row of pans from bottom to top, and then passes out through a small stack.

Hydraulic Press.—This is of Watson and Stillman's make. It is set at 1000 pounds and pure glycerine is used for the fluid. The mold is 6 inches in diameter and 4 inches high. The pressed cakes are about 1½ inches thick and weigh 60 to 70 ounces.

Melting-Furnace.—This is of the ordinary type for 2 No. 50 black-lead crucibles and provided with 3 dust-chambers.

Pulverizer.—A small Brückner ball-mill, 800 mm. in diameter and 500 mm. in width, with 40-mesh screens, is used for pulverizing the slag, etc.

Pumps.—Two No. 2 Korting hard-lead syphon-pumps with

platinum nozzles are used to handle the solutions. Pumps made of all hard lead from the Hanover works have given better satisfaction than those enclosed in an iron shell.

Installation.—The plant was installed in the building erected for the Stetefeldt process of refining, and as much as possible of the old plant was utilized in the installation. Necessarily, therefore, the arrangements are not as convenient and satisfactory as they might be. The main building is 52 by 42 feet with an annex 52 by 25 feet in size; but most of the annex is taken up for space in handling the sulphides before they are turned over to the refinery, and the silver-dryer stands outside the building. Recently a new melting-room with hearths has been erected.

PROCESS.

The process consists in boiling the sulphides in strong sulphuric acid to convert the sulphides into sulphates. The sulphate of silver is soluble in strong sulphuric acid, but the anhydrous sulphate of copper is practically insoluble. Owing to the large percentage of copper in the sulphides a large quantity of insoluble sulphate is produced, and this is one of the most serious difficulties of the process. After boiling, the charge is removed to the dissolving-tank, into which are put pure water, wash-water and weak solutions. Here the copper sulphate goes into the solution along with the sulphate of silver. The solution is filtered into the precipitating-tanks, where the silver is precipitated by metallic copper, after which, when strong enough, the solutions go to the crystallizers to recover the bluestone. Periodically the cement-silver is removed to the filter, sweetened, dried, pressed and melted. The mother-liquors are concentrated and crystallized, and the recovered acid is finally sent back to the pot. The residue in the dissolving-tank is taken out, washed somewhat and re-boiled, to recover as much as possible of the silver that it contains.

PRACTICAL OPERATIONS.

The first charge of the 1894 sulphides was put into the pot February 20, 1894, and the first charge of the 1895 sulphides was started February 27, 1895, so that the run was a few days over a year. A part of this time was taken up in the annual clean-up. A charge of 975 pounds is put into the pot in the

morning with about 1000 pounds of acid (66°) and thoroughly mixed and the charge heated. At first the reaction is rather violent. SO_2 is copiously evolved and the fumes carry considerable S, which gives them a yellowish color. At this stage the steam-jet may be required to increase the draft. After awhile the reaction settles down and the normal charge boils quietly until near the end. As soon as the charge gets stiff, more acid, about 100 pounds, is added, until about 3000 pounds have been used. Toward the end, evaporated acid is used. The strong acid is first drawn from the overhead receivers into a cast-iron pan, an old clean-up pan from the amalgamating-mill, holding about 180 pounds, through a stone-ware cock. It is then run into the pot through a rubber hose.

As the boiling goes forward, the anhydrous sulphate of copper is formed in large quantities, which separate, forming granular masses. This necessitates frequent stirring of the charge; and this, in turn, is severe on the pots.

The progress of the operation can be watched by taking out a small sample of the charge, treating with water and adding HCl to the solution; but this is not necessary after getting familiar with the process, since the color changes from black to brown or gray. About 90 per cent. of the total acid used is added before the charge begins to show soluble silver salts. Then the charge foams violently and must be constantly stirred, while the fire must be lowered. In about an hour the foaming is over and the charge is finished. This usually occurs in the afternoon of the day after starting.

The boiling is done in day-shifts only. The charge could now be removed to the dissolving-tank, but it is too thick, from the large amount of separated sulphate of copper, to be syphoned out; so it would have to be dipped out with a ladle. As it is now very hot and giving off SO_2 freely, this would be a disagreeable operation.

On the second morning, therefore, the charge is warmed up, generally with the addition of some acid, until it is sufficiently fluid, when it is ladled out into the sloping trough, which delivers it to the dissolving-tank. The pot is then started on a new charge. Although a charge could be turned in less than 2 days, this makes a very convenient and generally economical arrangement.

When starting the charge and when finishing it, the pot requires much of the potman's time; but at other times it does not call for much attention. The potman attends also to the dissolving- and the filtering-tanks, and to the precipitation of the silver. The ladle used in emptying the pot is a wrought-iron scoop and holds about half a gallon.

The dissolving-tank is filled with cold water to within 6 or 8 inches of the top and tightly covered, since the introduction of the charge generates much heat. After the charge is in, the cover is raised and the solution is stirred with a wooden paddle and boiled with steam, after which it is settled about half an hour and drawn into the filters. The first tankful of solution contains most of the copper, and is run into the small precipitating-tank and kept separate from the rest of the solution. After the precipitation of the silver it contains, it is run directly to the evaporators, and brought up to 35° B. and then crystallized.

The charge now resembles thick white mud and is washed from 8 to 10 times with weak acid solution, to remove the silver, yielding solutions about 20° B. in strength.

After washing, the residue is thrown into a filter. This residue varies very much, running from 5,000 to 19,000 ounces of silver per ton and 50 to 100 ounces of gold, the balance being mainly sulphate of lead. Some of the silver is present as sulphate, but there is also considerable metallic silver.

The solution has a reducing action, and immediately a separation of metallic silver begins in the dissolving-tank, often with the formation of beautiful growths upon the surface of the liquid. This reaction continues in the filters. By reason of it, metallic silver is found in the first residue, and some 10,000 ounces may accumulate in the filters during the year's run. In the annual clean-up, the sand is removed from the filters and boiled in the pot with strong acid to recover the silver. A small amount of silver and some gold remain in the sand.

From the filters the silver-bearing solution goes to the precipitating-tanks, where the silver is precipitated by copper, cathode-plates from an electrolytic refinery being used. The plates are stood up against the sides as close as possible. The precipitated silver falls to the bottom and is scraped away from the plates every few days with a wooden shovel. The copper

solutions require a long time to precipitate, sometimes 18 hours, but ordinary solution is precipitated in 4 to 5 hours. During the precipitation the solution is stirred by air and heated by steam.

When the precipitating-tank is cold and the hot solution of silver sulphate runs in, there may be a separation of silver sulphate, which may go into solution again as the solution is heated up, but some of it may also remain in the cement-silver, and be removed in washing the silver, in which case the wash-water must be treated with copper. This is done in the wash-solution storage-tank.

When about 20,000 ounces of cement-silver have accumulated in the precipitating-tanks, it is shoveled out into the sweetening-tank with a wooden shovel, washed with hot water and then with acidulated hot water until the ammonia-test shows no copper. This takes about 15 hours. The wash-water runs through a guard-tank containing scrap-iron, and then to waste. The sweetened silver is put into iron pans and dried about 24 hours in the steam-dryer, pressed into cakes, dried again and melted. A small copper shovel and an ordinary wisp-broom are used in handling the cement-silver at the press.

In 1894 the melting was done in graphite crucibles, holding about 2400 ounces or 2 bars each, coke and charcoal being used for fuel. The crucible was filled with cakes and a little borax was added. As the cakes melted down, more were added with a pair of tongs, until the crucible was nearly full of molten silver. Niter was then added, and sometimes a little borax, and the crucible was stirred with an iron rod, after which the slag was removed with an iron skimmer. This was repeated until the surface became clear and bright. After three or four additions of niter, the metal begins to boil. The chief impurity in the cement-silver is the iron rust from the pans, which is easily removed in the slag. About 1.4 pounds each of borax and niter are used for each crucible-charge. The cost of melting was about 0.1 cent per gross ounce.

When the slag has all been carefully removed, the metal is stirred and sampled. The crucible is then hoisted out of the furnace and the metal is cast. The melted silver is poured into heated and greased, light cast-steel molds. After pouring, a little sugar is thrown on the liquid silver and the mold is covered by a tight-fitting cast-iron cover. This gives a very

smooth surface to the bar. When cool the bars are hammered up and marked. The average fineness for 1894 was 999.4 silver, with no gold. As already stated, 446 bars were produced, of which 401 were 999.5 fine and 45 were 999 fine. The ashes and slag were ground and the buttons of silver separated, after which they were sent to the smelter.

The dissolving-solution is sent back to the dissolving-tank, after the precipitation of the silver, and is thus used again and again. It gradually increases in bulk, from condensed steam, and the copper contents increase. When it reaches 20° to 25° B. it is filtered and run into the evaporators and brought up to 35° to 37° B. It is then run into the crystallizers and left 2 days, if there is time, which separates most of the blue-stone.

The solution then goes back to the evaporators, and is brought up to about 42° B., and crystallized again. This crop of crystals contains a great deal of sulphate of iron. After the removal of the solution the crystallizer is filled with cold water, which dissolves most of the iron with but little of the blue-stone. This solution is run to waste through the guard-tank and the blue-stone remaining is shoveled out. This removes most of the iron from the blue-stone without the loss of much blue-stone. This product is not as good as the rest, the crystals being very small, but it answers as well in preparing "extra solution" in the leacher.

The main solution is again evaporated to 50° to 52° B., and run into the crystallizer, where it is allowed to stand several days, to separate as much as possible of the iron it contains. It is then pumped to the storage-tanks above, and used in the pots. Considerable iron is deposited in the evaporators during the second and third evaporations. Periodically they are washed out clean with water and the solution is run to waste through the guard-tank.

After boiling five charges of sulphides about 750 pounds of wet residue are obtained. This is put into the pot and boiled with a little more than its own weight of acid, after which it is washed and the final residue dried. This residue is very complex in composition, although it is mainly sulphates of lead and silver.

Various attempts were made to analyze this material so as to show how the different substances were combined and distributed. Finally two samples were put through a course of

analysis, which, it was thought, would yield interesting and valuable results. The results obtained are interesting and valuable too, but they cannot be considered as harmonious or even satisfactory. I give the actual results obtained. I have spent considerable effort in trying to figure them out to a satisfactory conclusion, but have not been able to do so.

Two samples were examined. They were first treated with ammonia solution and filtered. The filtrate was neutralized with acetic acid, giving a precipitate of AgCl, 3.40 per cent. in one and 1.55 per cent. in the other. HCl was then added to the second filtrate, giving 6.14 and 1.60 AgCl respectively, equal to 4.62 and 1.20 per cent. of Ag, or 6.67 and 1.74 per cent. of Ag_2SO_4. BaCl was then added to the third filtrate, giving 16.89 and 16.52 per cent. H_2SO_4, of which 2.09 and 0.54 per cent. corresponded to the amount of Ag_2SO_4 found, leaving 14.80 and 15.98 per cent. possibly existing as free acid, or in some other combination. There is a certain quantity of free acid present, but it is hardly possible that all this acid could be free, since the samples are dry powders. A separate attempt was made to determine the amount of free acid present; but the difficulties in the way rendered this somewhat unsatisfactory. The best determinations indicate that one sample contained about 2.27 per cent., and the other 0.59 per cent. of free H_2SO_4.

The filtrate from the $BaSO_4$ was treated with H_2SO_4, and 0.48 and 0.60 per cent. of $PbSO_4$ were obtained. The final filtrate was treated for copper, and showed 0.46 and 0.16 per cent.

The residue was next treated with $HNaCO_3$ to decompose $PbSO_4$. Silver was not found in this solution. 22.95 and 18.10 per cent. of H_2SO_4 were obtained. The determinations of the lead showed that 19.82 and 17.62 per cent. of this acid were combined with lead to form normal $PbSO_4$, leaving 3.13 and 0.48 per cent. in some other combination. In the filtrate from the $BaSO_4$, 0.18 and 0.57 per cent. of $PbSO_4$ were found.

The residue was next treated with acetic acid, and the following results were obtained:

	I.	II.
$PbSO_4$,	61.12	53.93
Fe_2O_3,	0.45	1.22
CaO,	3.35	3.32
MgO,	0.03	0.04

Ag was not found in this solution.

The residue was next treated with nitric acid, and HCl added to the filtrate, giving 7.96 and 19.69 per cent. of AgCl, equal to 5.99 and 14.82 per cent. of metallic Ag. Most of this Ag was probably present in the metallic state, but there was also a little unaltered sulphide of silver present, and possibly other combinations of silver.

This solution also gave the following figures:

	I.	II.
$PbSO_4$,	0.94	1.45
Fe_2O_3,	0.40	0.28
CaO,	0.08	0.06
MgO,	0.03	0.05

The residue was again treated with ammonia, and yielded 0.40 and 0.30 per cent. of AgCl, and 0.09 and 0.02 per cent. of $PbSO_4$. The final insoluble residue after all this treatment was 5.78 and 5.26 per cent.

Separate portions were treated with Cl in alkaline solution, and yielded the equivalents of 39.32 and 34.08 per cent. of H_2SO_4.

The following table gathers up the results. The sulphuric acid is stated as H_2SO_4, but only a small portion was actually present as such:

Final Residue.

	I. Per cent.	II. Per cent.
Ammonia solution:		
AgCl present as such,	3.40	1.55
AgCl, combination questionable,	6.14	1.60
H_2SO_4,	16.89	16.52
PbO,	0.35	0.44
Cu,	0.46	0.16
Bicarbonate of soda solution:		
H_2SO_4,	22.95	18.10
PbO,	0.13	0.42
Acetic acid solution:		
PbO,	44.98	39.69
Fe_2O_3,	0.45	1.22
CaO,	3.35	3.32
MgO,	0.03	0.04

	I. Per cent.	II. Per cent.
Nitric acid solution:		
AgCl, combination questionable,	7.96	19.69
PbO,	0.69	1.06
Fe_2O_3,	0.40	0.28
CaO,	0.08	0.06
MgO,	0.03	0.05
Second ammonia solution:		
AgCl, combination questionable,	0.40	0.30
PbO,	0.07	0.01
Final residue,	5.78	5.26

The final residue and the earthy constituents undoubtedly came from the sand of the filter gathered up in handling the material in the refinery.

During 1894 this residue was shipped away to the smelters for treatment, but now it is being melted on a hearth at the refinery.

SUPPLIES.

Sulphuric acid of 66° B. is purchased at Denver, Colo., and shipped to the refinery in iron tank-cars, holding from 40,000 to 50,000 pounds. It is drawn or syphoned out of the cars into a lead-lined iron tank on a truck and hauled to the refinery, where it is discharged into the storage-tanks by means of compressed air. It is necessary to have this truck-tank lined with lead, for the reason that, on standing, the small amount of acid necessarily left in the tank will absorb moisture and become dilute enough to attack iron. The handling of the acid requires some care, but presents no difficulty. It would be better if the acid could be blown by compressed air directly from the tank-cars to the storage-tank; and this arrangement may be made. In the 1894 run 389,439 pounds of 66° acid were used, an average of 3.34 pounds per pound of sulphides treated, or 0.68 pound per ounce of silver.

The particular form of copper used is of no consequence, provided it does not contain impurities that will reduce the fineness of the silver. At first we used cast-copper-plates, but afterwards used cathode-plates from an electrolytic copper-refinery. They were used direct without any preparation. When a pig-copper sufficiently free from gold can be had, it would be cheaper, especially if it carried silver, which would be recovered in the process.

In the 1894 run 16,832.5 pounds of copper were used. One

pound of copper precipitated 2.27 pounds of silver, or 33.1 Troy ounces. This consumption of copper is larger than theory requires, but probably some copper is oxidized by the air used in stirring the solution.

The local coal from the neighboring Weber field is used. Other and more expensive kinds have been tried, but do not seem to present any advantages. In the 1894 run 166,000 pounds were used for all purposes.

Labor.

For the ordinary operation of the process a superintendent and two men were required. Besides the general supervision of the work, the superintendent did the melting of the bullion about once in twelve to fourteen days, and required the assistance of an extra man. One laborer had charge of boiling the sulphides in the pot and the dissolving, filtering and precipitation of the silver. The other man had charge of the blue-stone and pressing the cement-silver. He also did the lead-burning, not only for ordinary repairs, but in the construction of the plant. Occasionally extra labor was required, particularly in shipping residue and making the annual clean-up. Mechanics were also required for special work at times.

Returns.

It is well known that there is a loss in determining the precious metals, particularly silver, by fire assay, arising from absorption by the cupel and slag. In the case of ordinary ores the amount of this loss per ton of ore is generally small, although the percentage of the total silver in the ore be large. In the case of rich materials, however, the percentage loss on the total silver is low, but the actual quantity per ton becomes considerable, and when the rich material carries copper, the loss of silver per ton becomes quite serious.*

In our business transactions the sulphides are always settled for upon an assay corrected for slag- and cupel-absorption as follows:

"Weigh out one-twentieth ($\frac{1}{20}$) of an assay-ton of sulphides,

* See "The Accuracy of the Commercial Assay for Silver," by the writer *Jr. Am. Chem. Soc.*, vol. xvi., pp. 505 to 516; also "The Inaccuracy of the Commercial Assay for Silver, etc.," C. A. Stetefeldt, *Trans.*, xxiv., 530 to 538.

55 grammes of granulated test-lead and 2 to 3 grammes of fused borax.

One-half of the lead is put in the bottom of the scorifier and hollowed out; the sulphides are put into the hollow and the rest of the lead poured over them; the borax is then placed on top. The assay is then conducted in the usual way. The slag and cupel shall be ground up and assayed, and the results added to the main assay."

This assay shows from 100 to 200, or even more, ounces per ton more than the ordinary uncorrected assay shows.

Even on the corrected assay the actual amount of silver returned by the refinery on the year's work was 2,078.81 ounces more than the assays called for. Mr. Russell, who has had large experience in this department of assaying, has declared to the writer that, in his opinion, the very best assay of Russell sulphides that can be made, shows still about $\frac{3}{4}$ of 1 per cent. below the actual amount of silver present. Of course, there must necessarily be some loss in our practical operations, but these returns show that this loss is less than the difference between the corrected assay and the amount of silver actually present. It is regarded as an extraordinary showing for a chemical process on the large scale to recover more than the best possible assay calls for.

The sulphides treated, 116,519.5 pounds, contained 572,544.45 ounces of silver; and the silver returned was divided as follows:

Product Returned.

	Fine silver. Ounces.	Total silver. Per cent.
Fine bullion, free from gold,	551,329.89	96.29
Residue,	15,773.41	2.76
Cleanings,	5,328.87	.93
On hand,	2,191.09	.38
Total,	574,623.26	100.36
Plus clean-up,	2,078.81	.36

All weights of sulphides and products, excepting one case covering less than 200 ounces, and all the assays are the originals made by the Daly Mining Company.

On selling the silver the reclamations by the buyers amounted to only 130.6 ounces for the whole year.

As to the recovery of the gold, I cannot see any reason why it should not equal the silver recovery, but the figures upon this point are not satisfactory. The actual return of gold for the year was 606.9 ounces, the original assays of the Daly Mining Company called for 654.8 ounces, but their re-assay on some of the samples reduced this to 646.1. This left an apparent shortage in the returns of 39.2 ounces. The same samples were assayed by Mr. Charles Earl, under my direction, and while the silver results showed a satisfactory agreement with the Daly assays, yet his gold determination called for only 602.9 ounces, showing a plus clean-up on the year's work of 4 ounces. After the close of the year's business, a general sample was prepared by taking proportionate weights of each of the check samples of the 25 lots, and the Daly Company's assay of this sample called for 605.9 ounces, and showed a plus clean-up of 1 ounce. Mr. Earl is no longer with me, so I cannot give his figures on this sample.

There are special difficulties in determining such small quantities of gold in the presence of so much silver. The Daly Company's assayer assayed the same samples of 3 lots of sulphides at three different times with the following results:

Gold—Ounces per Ton.

1st assay, ...	10	9.6	9.6
2d "	9.7	9.8	9.2
3d "	9.6	9.4	9.0

Mr. Earl found in the same samples 8.8, 9 and 9 ounces respectively.

These assays came on a run of about 24,700 pounds of sulphides, and the difference between the Daly's first and last assays changed a minus clean-up to a plus clean-up.

Again, the Daly Company assayed a sample of sulphides at two different times, and Mr. Earl assayed the same sample at two different times, with the following results:

Gold—Ounces per Ton.

	Daly.	Earl.
1st assay, ...	13.4	11
2d "	12.0	11.06

The difficulty of determining the gold is not confined to the

sulphides. The Daly Company's assays of 2 samples of gold bearing residue in triplicate were as follows:

Gold—Ounces per Ton.

1st assay,	143.4	118.6
2d "	137	124.2
3d "	141.1	126.2
Average,	140.5	123

Another average on the first sample was 138.1. In one ship ment of this residue the smelter paid for 0.48 ounce of gol more than the Daly Company's assay called for.

It has been suggested that the silver bars were not entirel free from gold.* While this was the case in the early days o the process, yet the assayer did not report any gold in any bar of silver shipped in 1894. With his previous experience in thi matter, and with the care used in looking for gold, it is hardl possible that any considerable quantities of gold would hav slipped out in the silver. A few ounces might possibly hav escaped, but hardly 39 ounces.

The conditions of the process are such that I do not see hov we could gain so much on the silver and lose on the gold; s that I am satisfied that the process practically recovers all th gold that goes into the operations, although the assays may no always show this.

The blue-stone product amounted to 175,809 pounds, or 3.6 pounds per pound of total copper going to the refinery. Thi copper includes the copper in the sulphides and the copper use to precipitate the silver. The return was somewhat belov theory, but an unknown quantity of copper was recovered i the guard-tank, through which all the solution is passed befor going to waste.

No particular care is taken to prepare fine large crystals c blue-stone; and it is not necessary to purify the solutions fro iron except as above described. Most of the blue-stone pr duced goes to the leacher, and the size of the crystals is of n moment whatever, while the small amount of iron present doe no harm. The best grade showed 0.34 per cent., the mediu

* Stetefeldt, *Lixiviation of Silver-Ores with Hyposulphite Solutions*, 2d Ed., p. 181

0.69, and the worst, of which only a small quantity was produced, 3.89 per cent. of protoxide of iron.

About 125,000 pounds of blue-stone were used by the leacher in preparing extra-solution, leaving about 50,000 pounds to be sold to outside parties.

Summary of Statistics for the Year of Boiling Russell Sulphides in Strong Sulphuric Acid by the Dewey-Walter Process.

First charge of 1894 sulphides to pot, February 20, 1894.
" " 1895 " " " 27, 1895.

	Pounds.
Sulphides treated,	116,519.5
	Ounces.
" contained silver by corrected assay,	572,544.45
" " silver average ounces per ton,	9,827.44
	Pounds.
" " copper,	31,585.3
" " lead,	385.6
	Per Cent.
" " copper average,	27.17
" " lead, "	0.33
	Pounds.
Acid used,	389,439.
" per pound sulphides,	3.34
" " ounce silver,	0.68
Coal,	166,000.
Copper used to precipitate silver,	16,832.5
Total copper,	48,417.8
One pound copper precipitated silver,	2.27
Blue-stone produced,	175,809.
" " per pound copper,	3.63

Regular labor, superintendent and 2 men.

Extra labor, 1 man at bullion melting to assist superintendent, laborers for clean-up and shipment of residue, etc., mechanics for special work occasionally.

116,519.5 pounds sulphides contained 572,544.45 fine ounces of silver.

Silver Product Returned:

	Fine Ounces.	Percentage of Total.
Fine bullion, free from gold,	551,329.89	96.29
Residue,	15,773.41	2.76
Cleanings,	5,328.87	0.93
On hand,	2,191.09	0.38
Total,	574,623.26	100.36
Plus clean-up,	2,078.81	0.36

The advantages of this process are:

1. It recovers a phenomenal percentage of the silver.

2. It is an entirely liquid process from the beginning to the end, so that there is no loss from handling dry products.

3. There is no roasting to cause loss.

4. A large percentage of the silver is recovered as very fine bars, ready to enter the market.

5. The process is so simple and so easily carried out, and the plant is so small and inexpensive, that it can be installed at individual leaching-works.

6. Finally, the cost of operation is small; in fact, the value of the blue-stone recovered returns a large proportion of the operating-expenses.

Laboratory-Note on the Heat-Conductivity, Expansion and Fusibility of Fire-Brick.

BY J. D. PENNOCK, SYRACUSE, N. Y.

(Colorado Meeting, September, 1896.)

THE different samples of brick examined were Grecian magnesite, American magnesite, silica brick and coke-oven tiling made in Belgium and used in retort coke-ovens.

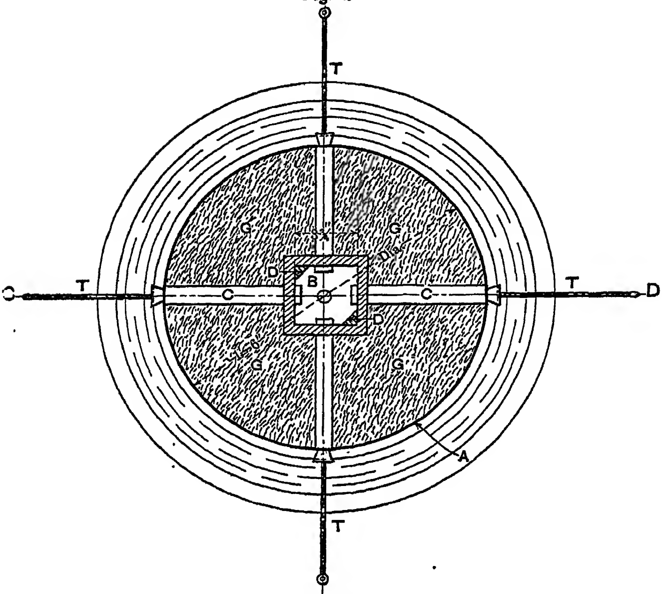
The Grecian magnesite was furnished by a New York party.

The American magnesite and the silica brick were furnished by a manufacturing company in Pittsburgh.

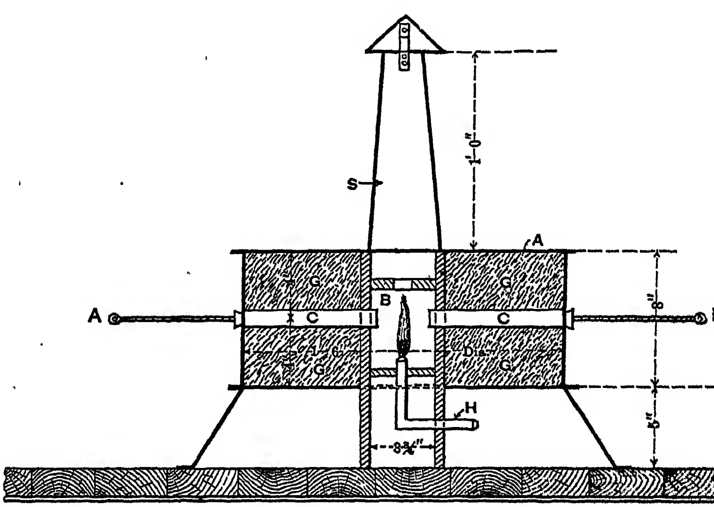
From these brick were made cylindrical cores, 7¾ inches long and 1⅝ inches in diameter. At one end of each was drilled a hole about ¼-inch in diameter and 1¼ inches long, into which the thermometer was inserted during the experiment.

Conductivity.—The apparatus used for determining the conductivity, as shown in Fig. 1, is a sheet-iron drum, 18 inches in diameter and 8 inches high. In the center is a little combustion-chamber with walls of split fire-brick, 3¾ inches square and about 6 inches high. The fire-brick, however, pass through the drum to the table on which the drum stands. The cores are placed horizontally, half-way between the top and bottom of drum and on the line of diameters at right angles. In the four walls of the combustion-chamber are cut holes just large enough to allow the cores to pass through. The cores extend

Fig. I.



SECTION ON LINE A B

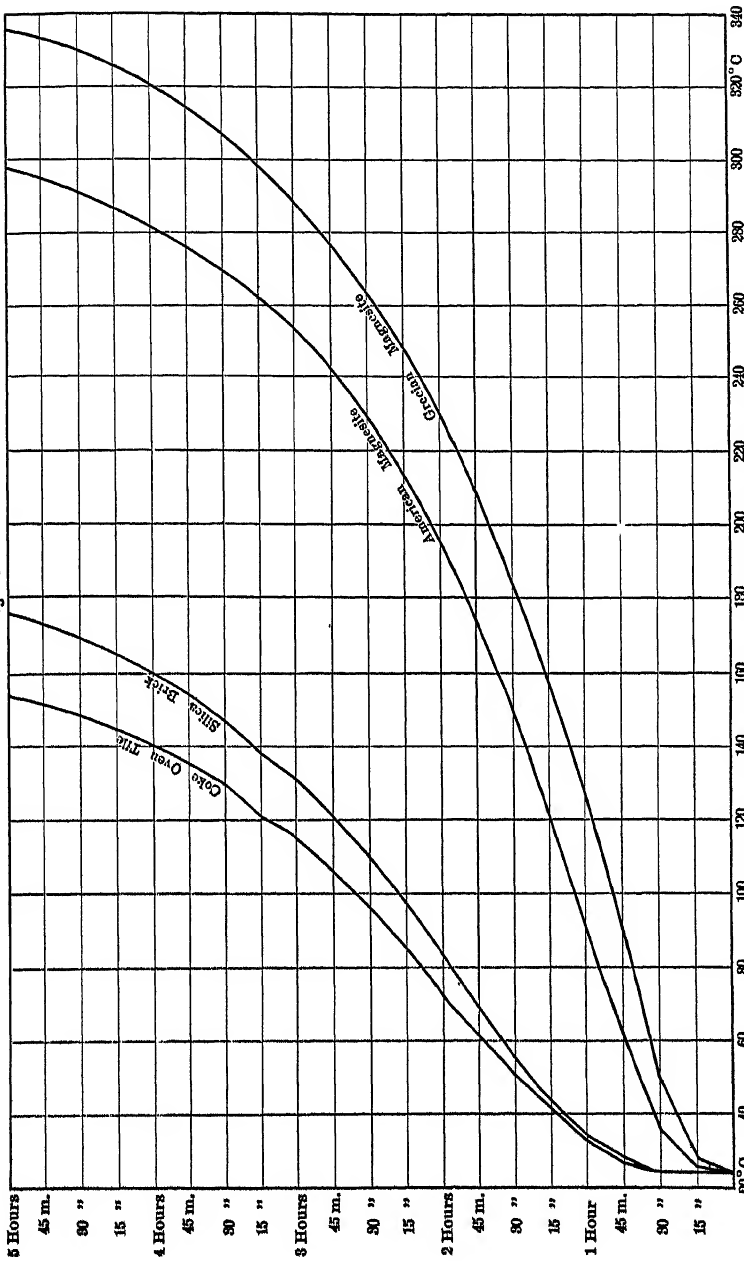


SECTION ON LINE C D

Apparatus for Determining Heat-Conductivity.

A, sheet-iron drum; B, combustion-chamber; C, cores to be tested; D, supports for Seger cones; G, glass-wool; T, thermometers; H, blast lamp for heating; S, chimney for combustion-chamber.

Fig. 2.



Diagram, showing Rate of Increase of Temperature.

into the combustion-chamber exactly ¼-inch. The other end of the core just comes to the outside of the drum through a hole. An asbestos washer encircles the thermometer, which is inserted in the brick, and covers the end of the core. A blast-lamp is introduced into the bottom of the combustion-chamber.

The space about the cores in the drum was filled with glass-wool, packed closely. The thermometers were read at the time the heat was started and every 15 minutes for five hours. Below are given the results of three experiments, and a chart of one of them may be found on the preceding page.

Temperatures of Ends of Cores at Stated Intervals.

(In Centigrade Degrees.)

Experiment No. 1.

Time. Hrs. Mins.	Grecian Magnesite.	Silica Brick.	Coke-Oven Tiling.	American Magnesite.
00,	25	24	24	25
15,	28	24	24	26
30,	49	24	24	36
45,	87	28	27	61
1 00,	125	34	33	92
1 15,	156	43	41	121
1 30,	184	54	50	149
1 45,	208	68	61	172
2 00,	230	82	73	196
2 15,	249	97	85	215
2 30,	264	109	96	229
2 45,	277	120	107	243
3 00,	289	131	115	254
3 15,	299	138	121	262
3 30,	308	148	130	270
3 45,	314	154	135	276
4 00,	320	160	140	281
4 15,	325	165	144	283
4 30,	330	169	148	290
4 45,	332	173	151	294
5 00,	337	177	154	297

Experiment No. 2.

00,	20	20	20	20
15,	27	22	20	26
30,	54	23	24	44
45,	86	25	27	69
1 00,	130	33	35	107
1 15,	157	40	42	132
1 30,	188	52	50	159
1 45,	215	63	59	184
2 00,	249	78	73	210

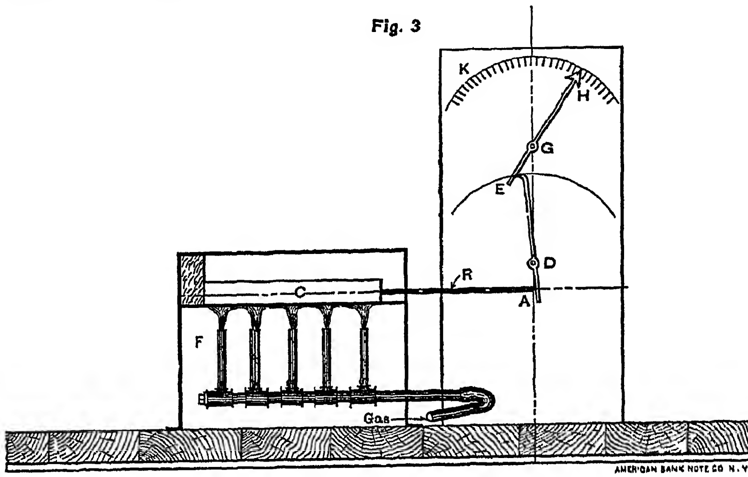
Time. Hrs.	Mins.	Grecian Magnesite.	Silica Brick.	Coke-Oven Tiling.	American Magnesite.
2	15,	263	93	85	229
2	30,	280	107	97	246
2	45,	295	121	110	260
3	00,	307	132	120	274
3	15,	317	142	130	.....
3	30,	325	152	138	294
3	45,	332	160	146	301
4	00,	339	166	152	307
4	15,	341	170	157	311
4	30,	341	175	162	315
4	45,	343	178	165	316
Experiment No. 3.					
	00,	25	25	25	25
	15,	29	25	25	29
	30,	50	26	26	43
	45,	83	28	29	69
1	00,	130	35	35	105
1	15,	157	41	42	130
1	30,	196	55	54	169
1	50,	220	71	70	205
2	00,	243	77	75	215
2	15,	263	90	87	236
2	30,	280	104	100	257
2	45,	297	117	112	270
3	00,	307	126	120	282
3	15,	317	139	132	294
3	30,	326	148	142	304
3	45,	334	167	159	319
4	00,	340	174	165	324
4	15,	345	180	171	325
4	30,	351	186	176	329
4	45,	354	191	182	329

The chart (Fig. 2) shows more strikingly the greater conductivity of the Grecian fire-brick. It will be noticed also that the Grecian magnesite conducts heat more readily than the American magnesite. This is doubtless due to its greater purity as well as its greater density. From the greater conductivity of Grecian magnesite, its advantage over the ordinary coke-oven tiling is apparent in cases where the aim is transmission of heat.

Expansion.—For the determination of the expansion of the fire-brick, an ordinary small combustion-furnace (F, Fig. 3) was used. As shown in Fig. 3, the fire-brick core, C, was placed in the furnace and forced and fastened against a fire-brick, which in turn was fastened against the iron end of the furnace, so that the

expansion was exhibited only at one end. Let into this end of the core was a glass rod, R, protected by an asbestos partition from the heat of the furnace. The other end of the glass rod touched the end of the short arm of a lever, A. The end of the long arm of the lever worked another lever at E. The levers are so arranged that the expansion of the core is magnified 70 times on the dial K.

Fig. 3



Longitudinal Section of Furnace and Apparatus for Measuring Expansion.

The following are the results:

	Length of Core. Inches.	Dial Reading. Inches.	Expansion of Core. In Inches.	Expansion per 12 inches of Brick.
Grecian magnesite, . . .	No. 1, 7¾	5	.07	.11
	No. 2, 7¾	5	.07	.11
American magnesite? . . .	No. 1, 7¾	4.75	.067	.10
	No. 2, 7¾	4	.057	.088
Coke-oven tiling, . . .	No. 1, 7¾	3.5	.05	.076
	No. 2, 7¾	3.5	.05	.076

The silica core broke, and we were unable to get a figure for its expansion.

Analysis of Cores.—Analyses were made of each material experimented with. The results were as follows:

	Grecian Magnesite.	American Magnesite.	Silica Brick.	Coke-Oven Tiling.
Silica, ...	2.16	3.10	94.07	69.89
Iron oxides and alumina,	72	6.64	3.66	27.75
Calcium oxide, . . .	4.20	3.76	1.39	0.27
Magnesium oxide, . .	93.03	86.50	0.19	0.17

Specific Gravity.—By displacement of water method the specific gravities of each fire-brick were:

Grecian magnesite,	3.54
American magnesite,	3.44
Coke-oven tiling,	2.56
Silica-brick,	2.54

Weight Per Cubic Foot.—Each core was weighed, and by calculation from these weights the following weights per cubic foot were obtained:

	Pound per Cubic Foot.
Grecian magnesite,	170.2
American magnesite,	160.9
Coke-oven tiling,	109.9
Silica-brick,	111.4

Fusibility.—Pieces of each fire-brick were put into the combustion-chamber of the gas-fired boiler, where the heat, as determined by Seger cones, was 1270° C., and allowed to remain; but there were no signs of softening in any one of them. The Grecian fire-brick, which, when put into the fire, was of a yellowish-brown color, was bleached to a perfect white.

We could not obtain a heat sufficiently high to fuse any of the fire-brick.

Action of Blast-Furnace Gases Upon Various Iron-Ores.

BY O. O. LAUDIG, BUFFALO, N. Y.

(Colorado Meeting, September, 1896.)

It is a well-established fact that some ores do not reduce as readily in the furnace as others, thus seriously affecting out-put, and consequently, cost of product. With the object of obtaining some information upon this subject, the following experiments were made in the laboratory of the Buffalo Furnace Company.

Efforts were made to obtain, as nearly as possible, representative ores from some of the most important mining districts in the United States. The ores received were from the Mesabi, Marquette, Gogebic, Menominee and Vermilion ranges of the Lake Superior district and from Southwest Virginia. Requests

for ores from Alabama failed to receive a response. To the above ores were added several samples of scales and cinders. It may be said in this connection that a number of the samples received were selected and do not exactly represent the average product of the mine, as will be seen by comparing the analyses with the amount of metallic iron found in the sample used in the experiments.

In order to classify the ores for the experiments, they were dried so as to sieve easily, and, without crushing, were sieved through ¼-inch, ⅛-inch, 20-, 40-, 60-, 80- and 100-mesh sieves. The percentage remaining on each sieve was determined by weight. The results are shown in Table I., under the head "Relative Fineness, Not Crushed."

After all the samples had been treated in this way, they were classified as nearly as possible, according to their size, as shown by the percentage which remained on or passed through a certain mesh.

Five divisions were thus made. Those samples which were all lump, or nearly so, were crushed in an iron mortar so as to pass through a 20-mesh sieve, but only the part remaining on an 80-mesh sieve was taken. The ores next in size were crushed through a 40-mesh, and that remaining on a 100-mesh was taken. The next two sizes were crushed so as to pass through 60-mesh and 80-mesh respectively, and the entire sample was taken. The finest ores were crushed through a 100-mesh sieve. In crushing, care was taken to sieve repeatedly, so as to have the least possible quantity pass through the smaller-meshed sieves. The object of this classification was to obtain for the experiment, as nearly as practicable, a relative size of the ores as shown by natural condition.

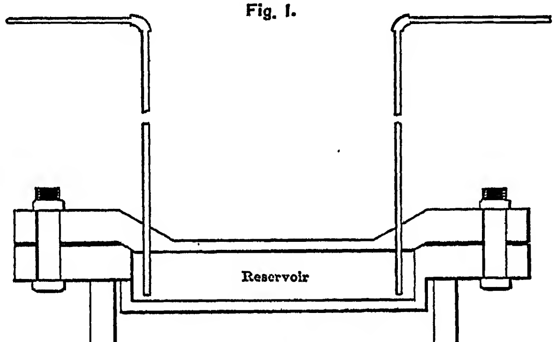
After crushing, the percentage remaining on sieves finer than the largest mesh through which the ore was crushed, was determined, thus giving some idea of the size of the particles of the ore. In cinders, only lumps were taken and treated with the coarsest ores. Three samples of No. 26 were taken, one crushed through a 100-mesh, another through 20 on 80, and the third through ⅛-inch on 20-mesh.

Table I. shows range and class of ore, size in natural condition and percentage remaining on various-sized sieves after crushing; also, the classification for experiments.

The magnetite No. 29 is from an unknown source, and was taken because of its large crystals; the other magnetite having very small crystals.

In the column headed "Cohesion, as Shown by Crushing," No. 26, a hard Minnesota ore, was taken as the standard for "Very Strong," and No. 30, a spongy, ocheriferous limonite, for "Very Weak," and the relative cohesion was estimated by the readiness with which the ore went to pieces when crushed in an iron mortar. Some ores, like the magnetite No. 29, were easily broken into small crystals, but these crystals were difficult to pulverize.

Fig. 1.



SECTION OF APPARATUS.
One-fourth natural size.

The total analyses given in Table II. are an average of the mines' product. All samples were dried at 212° F., and the specific gravity and metallic iron were determined in the samples used for the experiment.

The apparatus used (see Fig. 1) was a cylindrical cast-iron chest, made in two pieces, top and bottom. In the bottom was a reservoir, 6 inches in diameter and 1⅛ inches deep. A ground-joint was made by means of a ½-inch bevel on the upper edge of the reservoir and on a circular projection of the lid. Both lid and bottom had flanges 2¼ inches wide, in which were four holes ½-inch in diameter, used for bolting the two parts of the chest together. Passing perpendicularly through the lid and just within the circumference of the reservoir, was an ⅛-inch pipe which extended almost to the bottom of the reservoir. Directly opposite was a similar pipe, similarly situated, one being used as an inlet, the other as an outlet to the reservoir.

The whole apparatus stood on four cast-iron legs 1½ inches high, thus permitting any liquid in which the apparatus was immersed to circulate freely on all sides. To obtain a sufficient heat for the experiment, a lead-bath was used, the lead being melted in a steel vessel placed over a Fletcher burner. Before immersing the apparatus in the bath, fire-clay was put on the flanges of both top and bottom and the parts were tightly bolted together. About 10 inches above the lid of the apparatus an elbow was put on both inlet- and outlet-pipes of the reservoir, and the pipes were extended 2 feet beyond the outer edge of the steel vessel containing the lead, so as to prevent burning of any rubber connections used. On both of these pipes were placed 2 valves, 1 set being used for shutting off, and the other set for regulating, the current of gas passing through the reservoir.

In order to find the loss of the various ores, etc., by heating, ½-gramme of each crushed sample was weighed into a porcelain combustion-boat 1½ inches long, and, all being placed in the reservoir, the apparatus was sealed, bolted and immersed in the lead-bath, as described. Air, which was passed through caustic potash (sp. gr. 1.27), pyrogallate of potash and concentrated sulphuric acid, was then forced for three hours through the ⅛-inch pipe leading into the reservoir. The pressure was obtained from an ordinary laboratory blast-lamp connected with the cold-blast main of the furnace. In the lead-bath, zinc melted and antimony did not even become plastic during the entire heating. This gave an almost constant temperature of about 810° F. After three hours' heating, the apparatus was taken from the bath, closed by means of the valves in the ⅛-inch pipes and allowed to cool. Upon weighing, the various ores, etc., showed a loss per cent., as indicated in Table II., under "Lead-Bath Results." It was the intention to have no carbon dioxide, oxygen or water pass over the samples; but the gain in the scales, etc., (indicated by the + mark in the table) shows that some oxygen remained in the air, probably due to a saturated solution of pyrogallate of potash. After this heating, all the ores except the magnetites tended toward the red color of oxide of iron. They did not increase in bulk; on the contrary, many occupied less space in the boats than before heating.

The next experiment was with the furnace-gases. Two

water-tight barrels were used to collect these gases. Into the head of each barrel were put two quarter-inch pipes, one extending just within the barrel and the other almost to the bottom. Near the bottom of each barrel was another pipe, 1 inch in diameter. Valves were placed on each pipe. These barrels were filled with water and taken to the top of the furnace, and gas was forced into the barrels through the pipe extending just within the head, while the water flowed out of the inch pipe at the bottom. Having been filled with gas, they were taken into the laboratory. The pipe passing through the head and to the bottom of each barrel was used to admit water, thus forcing gas through the other pipe in the barrel-head, during the experiment.

Gases were drawn from the furnace at 10 feet below the stock-line. For this purpose a ½-inch pipe, 22 feet long, was allowed to settle with the stock, the gases being taken from it and collected in barrels, as described.

While taking the gases the furnace was full and showed 10 pounds pressure at the tuyeres. At the top of the furnace, the pressure was 2.35 ounces, and at the end of the pipe from which the gases were taken it was 10 ounces. Three engines were making 17 revolutions each per minute, representing 27,478 cubic feet of piston displacement.

The dimensions of the furnace are: Bosh, 18 feet; diameter of bell, 9 feet; height, 80 feet; diameter of crucible, 11 feet; stock-line, 13 feet; distance from nose to nose of tuyeres, 10 feet 1 inch.

There are 7 6½-inch tuyeres. The capacity from tuyere-level to stock-line is 12,078 cubic feet.

For the experiment with furnace-gases ½ gramme of each sample was weighed into a porcelain combustion-boat, and, as in the preliminary heating, the boats were placed in the reservoir of the cast-iron chest, the joints sealed and the whole immersed in the bath of molten lead (in which zinc melted and antimony did not during the entire heating). The gas in the barrels was then forced through the apparatus, the pressure being received by means of water flowing from an elevated vessel (a half-barrel) into the barrels holding gas.

The connections used were of rubber. The gases were heated in the ½-inch pipe by means of a Bunsen burner before

entering the reservoir containing the ores. A wash-bottle was connected with the outlet-pipe so as to give back-pressure. It showed 2.94 ounces. One-eighth-inch glass and rubber tubing was used throughout. The gases showed by analysis:

	First barrel. Per cent.	Second barrel. Per cent.
CO_2, ...	7.4	7.8
CO, ...	31.0	27.0
Proportion of CO_2 to CO, 1 to	4.19	3.46

In this heating 340 liters of gas were passed over the samples in two and three-quarter hours. The cast-iron chest was then taken from the lead bath and allowed to cool in a current of the gas used. At the end of the experiment the reaction was about as prominent as at any time during the two and three-quarter hours, as is shown by the following analysis:

	After 15 min. heating. Per cent.	After 45 min. Per cent.	After 1¼ hrs. Per cent.	After 2 hrs. Per cent.	After 2¾ hrs. Per cent.
CO_2, . .	10.6	9.8	12.0	13.4	13.0
CO, . .	28.4	26.2	22.0	18.6	18.8
Proportion, 1 to	2.68	2.67	1.83	1.83	1.45

The experiment was discontinued at this time for lack of gas. Upon opening the apparatus after it had cooled it was found that a number of the ores had increased in bulk, and had deposited so much carbon as to almost completely fill the porcelain boat, a bulk equal to four or five times that of the original sample. This will be discussed later.

The porcelain boats and contents were then weighed, thus giving the weight of ore and carbon deposited after treatment with gas. After determining the carbon its weight was deducted from the above, and, this result being taken from the original weight of the sample, the loss during the treatment with furnace-gas was obtained. From this loss was deducted the loss (in weight) in the preliminary heating, thus giving the weight of oxygen lost for ½ gramme of sample used. The percentage of oxygen lost was then calculated, based on its combination with metallic iron.

The carbon was determined by combustion in the usual way, all precautions being taken to purify the oxygen used and to convert any CO evolved into CO_2. For absorbing the CO_2 two

Geissler bulbs were used instead of one. In this way any saturation of the first bulb was noticed in time to prevent loss of CO_2, the second bulb taking up what passed through the first. The KOH in both bulbs was changed as soon as the carbonates began to separate in the first bulb.

The carbon deposited was very spongy and easily burned, twenty minutes being sufficient to burn, in a porcelain tube, as much as 0.18 gramme. Nearly all the ores, after being taken from the combustion-tube, showed a material increase in bulk, some occupying several times the space of the original sample. The scales, cinders and magnetites did not change in volume. No. 21 occupied less space after treatment than before.

It was noticed that the ores which deposited the most carbon increased most in volume, probably because of the carbon splitting up the particles and rendering them spongy. The amount of carbon deposited seems, however, to bear no relation to oxygen lost, as will be seen in Table II.

It may be well to say here that position in the reservoir had no bearing upon oxygen lost or carbon deposited, since several of the ores depositing most carbon were placed near the circumference of the reservoir and at right angles to a line passing through the outlet- and inlet-pipes. No. 30 occupied a similar position, yet it lost more oxygen than any other sample.

In the experiments three samples of No. 26 were used, principally because of the difficulty in always obtaining exactly the same conditions, thus giving a standard for any future treatment. The sample crushed through a 100-mesh sieve was an average of the ore, while the other two were taken from a hand-sample selected for its unformity and density. The three samples were placed side by side as near the center of the reservoir as possible, being subjected to practically the same treatment. All of these samples increased to about twice their original volume. They were also found to be decidedly spongy, although No. 26 is a dense, massive ore. The pieces of the coarsest sample (through $\frac{1}{8}$ on 20) were spongy on the outside, but contained small, hard grains in the centre, most of which passed through a 60-mesh sieve.

The other two samples were spongy throughout. This would indicate that the action was first upon the outside of the sample.

By referring to the column "Loss per cent. in Oxygen," in

Table I.

Number.	Range or Locality.	Class of Ore.	Physical Appearance.	Relative Fineness. Not Crushed. Percentage Remaining on Sieves. On ¼.	On ⅛.	On 20.	On 40.	On 60.	On 80.	On 100.	Thro 100.	Size of Particles as Shown by Crushing. Per cent. On 40.	On 60.	On 80.	On 100.	Thro 100.	Cohesion as Shown by Crushing.	Size of Ore as Used in Experiment.
1	Mesabi.........	Hematite.	Slaty structure.	24,41	7.07	11.24	10.58	11.60	0.20	6.90	28.00	......	......	......	......	......	Weak.	Through 100 mesh.
2	Mesabi.........	Hematite.	Compact lumps.	48.69	9.19	8.49	4.39	5.60	1.23	4.34	18.07	......	......	......	......	......	Strong.	Through 100 mesh.
3	Mesabi.........	Hematite.	Ocheriferous.	45.69	7.38	11.87	8.69	9.92	0.59	0.13	15.73	......	......	......	......	......	Strong.	Through 100 mesh.
4	Menominee.....	Specular.	Tending toward hematite.	47.09	7.20	15.00	5.20	3.16	0.68	1.36	20.31	......	......	......	......	......	Medium.	Through 100 mesh.
5	Menominee.....	Specular.	Small crystals.	29.17	18.99	15.46	5.30	3.52	0.95	1.47	25.14	......	......	......	......	......	Very weak.	Through 100 mesh.
6	Menominee.....	Specular.	Small crystals.	41.10	17.77	11.78	4.33	3.56	0.80	3.11	17.55	......	......	......	......	......	Weak.	Through 100 mesh.
7	....	Iron scale.	....	3.90	5.15	22.76	19.72	19.47	4.19	4.74	20.07	......	......	......	......	......	....	Through 100 mesh.
8	Gogebic.........	Hematite.	Granular.	67.72	8.58	6.29	4.02	4.28	0.53	1.22	7.36	......	......	......	26.93	73.07	Strong.	Through 100 mesh.
9	Menominee.....	Hematite.	Pebbly structure.	29.81	23.52	25.67	7.47	3.23	0.70	1.39	8.21	......	......	......	15.17	84.83	Medium.	Through 80 mesh.
10	Menominee.....	Specular.	Slaty structure.	52.57	20.36	13.69	3.69	1.99	0.46	3.69	3.55	......	......	......	9.95	90.05	Weak.	Through 80 mesh.
11	Menominee.....	Specular.	Slaty structure.	43.08	18.85	15.00	4.69	4.22	0.77	1.44	11.95	......	......	......	8.59	91.41	Weak.	Through 80 mesh.
12	Marquette	Hematite.	Granular.	17.42	16.94	37.32	9.85	5.46	1.49	1.82	9.70	......	......	......	37.84	62.16	Medium.	Through 80 mesh.
13	Menominee.....	Specular.	Slaty structure.	42.75	17.64	18.45	6.85	3.53	1.14	0.85	8.79	......	......	......	15.42	84.58	Weak.	Through 80 mesh.
14	Gogebic.........	Hematite.	Granular.	60.08	9.29	11.85	5.33	6.33	0.95	1.85	4.32	......	......	......	20.76	79.24	Weak.	Through 80 mesh.
15	Mesabi.........	Hematite.	Compact.	73.56	3.22	4.59	4.83	2.29	0.93	2.07	8.51	......	......	......	14.30	85.70	Weak.	Through 80 mesh.
16	Mesabi.........	Hematite.	Lumps dense.	49.32	13.98	21.82	5.94	3.73	0.32	0.78	4.11	......	......	15.29	18.15	66.56	Medium.	Through 80 mesh.
17	Marquette	Hematite.	Greasy.	57.70	17.06	13.37	5.30	1.80	0.90	0.71	3.16	......	......	14.23	21.89	63.88	Strong	Through 60 mesh.
18	Marquette	Hematite.	Greasy.	34.80	21.90	28.58	6.56	1.71	1.50	0.51	4.44	......	......	10.27	16.41	73.32	Medium.	Through 60 mesh.
19	Menominee.....	Hematite.	Granular.	99.50	0.50	(Selected sample.)						......	......	11.83	19.51	68.66	Medium.	Through 60 mesh.
20	....	Steel scale.	....	16.87	18.98	26.15	14.89	12.06	2.04	2.56	6.45	......	......	13.28	19.63	67.09	Medium.	Through 60 mesh.
21	Southwest Va..	Limonite.	Micaceous gangue.	100.00	(Selected sample.)							......	......	10.61	17.55	71.84	....	Through 60 mesh.
22	Marquette	Hematite.	Slaty structure.	60.72	20.38	11.58	2.79	1.48	0.25	0.30	2.00	......	40.56	6.14	17.48	35.82	Weak.	Through 60 mesh.
23	Vermilion	Hematite.	Dense.	66.07	15.46	12.81	2.17	1.28	0.18	0.35	1.68	......	45.34	34.15	3.03	17.48	Medium.	Through 40 on 100.
24	Southwest Va..	Limonite.	Spongy in part.	64.52	12.75	11.21	5.78	3.49	0.61	0.63	1.01	......	46.00	7.62	8.88	37.50	Strong.	Through 40 on 100.
25	Southwest Va..	Limonite.	Spongy in part.	81.09	13.81	3.97	0.52	0.27	0.05	0.07	0.22	......	44.09	6.81	9.59	39.51	Medium.	Through 40 on 100.
26	Vermilion	Hematite.	Dense, massive.	92.98	4.24	0.98	0.40	0.42	0.06	0.15	0.77	40.30	21.87	3.43	10.67	23.73	Weak.	Through 40 on 100.
26a	Vermilion	Hematite.	Dense, massive.	A selected lump.								......	......	......	......	......	Very strong.	Through 100 mesh.
26b	Vermilion	Hematite.	Dense, massive.	A selected lump.								......	......	......	......	......	Very strong.	Through ⅛ on 20.
27	Marquette	Magnetite.	Granular, crystals small.	Hand sample lumps.								17.18	11.36	3.56	11.05	56.85	Very strong.	Through 20 on 80.
28	Marquette	Hematite.	Dense, massive.	Hand sample lumps.								27.86	16.70	4.65	10.60	40.19	Weak.	Through 20 on 80.
29	....	Magnetite.	Granular, crystals large.	One large lump.								33.88	22.82	3.77	5.65	33.88	Very strong.	Through 20 on 80.
30	Tennessee	Limonite.	Ocheriferous, spongy.	Very small sample, selected.								32.08	20.82	4.43	7.52	35.15	Weak.	Through 20 on 80.
31	....	Flue cind'r.	....	Only lumps taken.								43.33	24.22	3.90	5.69	22.86	Very weak.	Through 20 on 80.
32	....	Puddle "	....	Only lumps taken.								47.59	21.25	3.39	4.75	23.02	....	Through 20 on 80.
33	Marquette	Specular.	Slaty structure.	Selected.								34.51	22.09	2.68	7.36	33.36	....	Through 20 on 80.

TABLE II.

Number.	Specific Gravity.	Met. Iron in Sample Used.	Lead-Bath Results.				Analyses.								
			Loss per cent. in First Heating.	Loss per cent. in Oxygen.	Per cent. of Carbon Deposit.	Carbon Deposit per unit of Iron.	Met. Iron.	Silica.	Sulphur.	Lime.	Alumina.	Magnesia.	Manganese.	Moisture.	Copper.
1	4.6992	64.35	0.34	29.37	17.78	0.2763	63.31	4 58	0.010	0.20	2.10	0.05	0.51	9.07	......
2	4.5454	61.63	2.00	17.83	10.20	0.1655	61.95	4.22	0.013	0.25	3.40	0.10	0.55	9.68	......
3	3.9401	61.88	4.84	26.69	22.08	0.3568	60.80	4.25	trace	0.71	0.93	0.07	0.49	9.97	......
4	4.6577	62.50	0.54	17.40	4.72	0.0755	63.37	4.53	0.003	1.50	0.72	1.90	0.19	....	......
5	4.7619	65.10	1.24	11.33	1.56	0.0239	65.10	4.08	0.022	0.19	1.25	0.37	0.39	7.60	......
6	4.7551	61.50	0.72	13.58	3.10	0.0504	61.00	8.29	0.027	0.36	1.98	0.40	0.36	8.10	......
7	4.5631	73.55	+1.30	0.09	0.09	0.0014	68.55	7.05	....	......	......	......	......	....	......
8	4.4643	60.24	1.10	26.76	23.60	0.3918	63.06	4.04	....	......	3.16	......	......	8.58	......
9	4.5935	60.44	0.08	11.89	6.96	0.1151	58.80	8.70	....	1.40	2.73	0.64	0.09	10.20	......
10	4.3554	55.81	0.90	15.80	1.98	0.0355	57.32	5.88	0.023	1.54	1.39	4.48	0.77	6.60	......
11	4.8614	65.85	0.26	18.35	4.66	0.0708	61.38	4.03	0.018	1.08	1.09	2.63	0.52	6.68	......
12	4.7449	63.89	0.30	13.07	4.64	0.0728	63.35	4.20	....	0.30	1.05	0.45	0.45	11.40	......
13	4.8722	65.98	0.08	15.00	2.50	0.0379	63.78	4.30	0.003	0.35	1.34	1.05	0.23	6.70	......
14	4.8135	65.20	0.90	25.13	35.22	0.5402	62.75	4.90	0.013	0.21	1.18	0.13	1.28	9.38	......
15	4.5499	64.80	2.80	25.93	36.40	0.5618	63.65	3.78	0.011	0.26	0.92	0.47	0.77	5.25	......
16	4.4813	61.03	1.46	19.08	7.42	0.1215	62.40	3.30	0.013	0.05	4.62	trace	0.67	10.00	......
17	4.5746	62.07	0.92	16.99	12.32	0.1985	61.00	5.62	0.012	0.53	1.32	0.54	0.70	11.85	......
18	4.5506	59.56	0.70	15.67	4.48	0.0751	62.77	4 66	0.021	1.15	1.17	1.01	0.37	10.20	......
19	4.7858	64.80	0.70	16.06	10.86	0.1671	60.00	4.25	0.020	1.80	1.84	0.42	0.60	8.18	......
20	4.0201	71.04	+0.30	4.07	0.62	0.0081	73.06	1.92	....	......	......	......	......	....	......
21	3.2801	46.74	8 94	11.98	0.98	0.0209	44.55	14.15	0.210	......	......	......	0.05	14.00	0.21
22	3.6566	45.64	2.10	15.95	12.60	0.2761	43.58	34.73	0.022	0.25	1.12	0.17	0.28	7.32	......
23	4.8839	66.24	0.64	6.26	3.24	0.0489	64.55	4.55	0.002	0.45	1.95	trace	0.04	6.19	......
24	3.7886	56.66	8.26	25.78	24.92	0.4391	53.35	7.48	0.100	0.82	2.37	0.49	0.05	8.00	......
25	3.4129	44.76	6.44	23.98	11.56	0.2582	45.00	13.00	0.200	......	......	......	3.00	9.00	......
26	4.8286	66.11	0.44	24.64	16.88	0.2533	67.33	1.67	0.010	0.98	0.61	0.21	0.03	1.00	......
26a	5.0633	68.22	+0.04	26.74	14.50	0.2125	....	....	....	......	......	......	......	....	......
26b	4.9355	67.81	+0.04	24.36	12.88	0.1898	....	....	....	......	......	......	......	....	......
27	4.2571	60.23	0.30	0.00	0.00	0.0000	66.50	4.67	0.040	0.64	0.07	0.25	0.23	....	......
28	4.9395	66.34	+0.10	13 22	2.16	0.0324	62.60	5.64	0.040	0.52	1.27	0.32	0.26	....	......
29	4.4743	58.68	0.04	0.00	0.10	0.0017	....	....	....	......	......	......	......	....	......
30	3.6935	59.50	9.78	50.04	14.30	0.2420	59.50	1.75	0.200	......	......	......	0.02	8.00	0.26
31	3.8827	47.60	0.08	0.00	0.08	0.0017	51.42	28.25	....	......	......	......	......	....	......
32	4.9103	57.13	0.30	4.57	0.74	0.0129	56.14	22.15	....	......	......	......	......	....	......
33	5.0251	67.94	0.15	24.84	16.88	0.2485	65.00	3.25	0.011	0.43	1.48	0.31	0.33	....	......

Table II., it will be seen that the size of the grains bears very little relation to the oxygen lost, inasmuch as the coarsest sample of No. 26 lost as much oxygen as the finest. This will make future experiments easier, as the sieving and classification of ores according to size is a very long and tedious operation.

In the amount of oxygen lost no distinct line can be drawn either as to class of ore, size or physical appearance, except in the case of magnetics, few of which lost any oxygen, several losing none. The hematites seem more easily reduced than the specular ores, yet No. 33, an almost perfect type of a specular ore, lost 24.84 per cent. of its oxygen, which is above the average lost by the hematites. Nos. 26 and 28 are both dense, massive hematites of about the same chemical composition and specific gravity, yet No. 26 became quite spongy, deposited about eight times as much carbon and lost twice as much of its oxygen as No. 28.

The preliminary sieving of the ores was purely experimental, as nothing definite was known regarding the effect of the size of the grains upon the amount of the oxygen lost. It was, however, thought that the finest ores would reduce easier than the coarsest ones. Reference to Table II. will show that this was not the case.

The writer wishes to acknowledge many valuable suggestions made by Mr. F. E. Bachman, manager of the Buffalo Furnace Company.

Excentric Jig, with Adjustable and Automatic Lower Discharge Arranged for the Full Width of the Bed and for One or More Compartments.

BY EDGAR G. TUTTLE, NEWARK, N. J.

(Colorado Meeting, September, 1896.)

The accompanying figures show the arrangement of a two-compartment excentric jig fitted with adjustable and automatic discharges for drawing off the lower product obtained in jigging minerals, ores, coal, etc., which is designed to remove or separate more effectively the materials treated, as they travel over the jig from the receiving- to the discharge-points.

The discharge-box is constructed to extend across the width of the jig at right-angles to the direction of the movement of the material over the jig-sieve, and to intercept at suitable points the path of the material and remove such of the lower product as has become separated, allowing the remaining material to continue on its passage to the next, and so on to the final overflow- or discharge-end.

Thus the discharges are more directly presented to the lower product, for receiving and removing it, than in the case of small discharge-boxes located at the center or side of the bed, or at the middle of the overflow-end; which arrangements require the lower product to follow a circuitous path before arriving at the discharge, thus consuming more time to effect a separation and lessening the capacity of the jig, or else permitting the material to pass beyond the discharges and out at the overflow without being separated from the upper product.

With jigs of a single compartment the wide lower discharge-box can be readily placed directly below the point where the upper product overflows from the jig; but in jigs of two or more compartments, the arrangement is usually a small discharge-box 3 inches square, more or less, or of other shape, located at the sides or in other positions of each compartment, as explained.

The arrangement shown in Figs. 1 and 2 permits of the introduction of an automatic and adjustable lower discharge at the end of each compartment of a jig, and can be applied to jigs with any number of compartments.

The construction and working of the jig fitted with a wide discharge-box is the same as that of other jigs, and the method of drawing off the lower product is similar, except as to the arrangement of the discharge.

The jig shown has two compartments, each consisting of two parts, in one of which is the jig-frame, K, and screen, L. The other part contains the plunger, A, operated through the rods, B, by the double adjustable excentric, C, on the shaft, D, which is driven by the pulley, E, by the side of which is the loose pulley, F.

With each revolution of the shaft, pulley and excentric the plunger makes a down- and then an up-stroke.

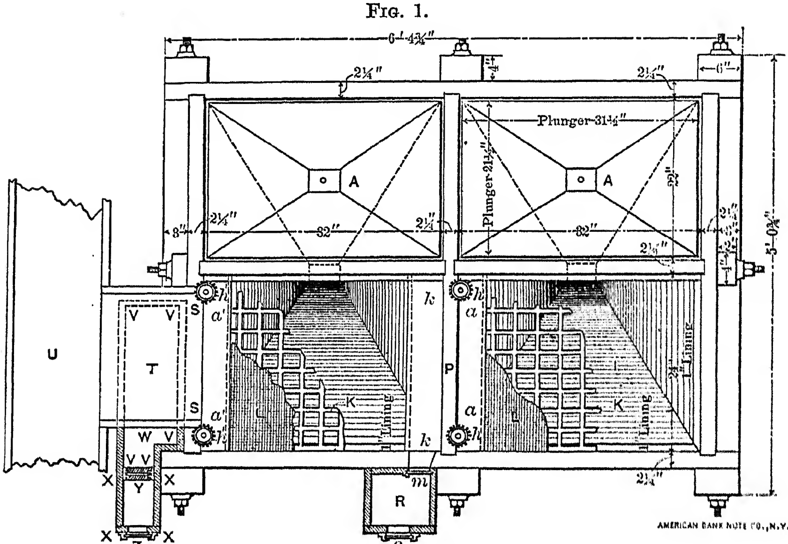
The jig is filled with water to the level of the overflow, J J, and the supply is regulated through the pipe by the valve, N.

The material to be jigged is fed upon the jig-bed through the spout, G, and hopper, M.

The motion of the plunger is communicated through the water in the box, or compartment, H, to the jig-bed, as in the usual process of jigging, thus tending to lift the specifically lighter particles up to the upper horizon of the layer of material on the jig-bed, while the heavier particles are not lifted as high, and remain nearer the bottom of the bed.

The particles thus treated, while travelling across the jig-bed

FIG. 1.



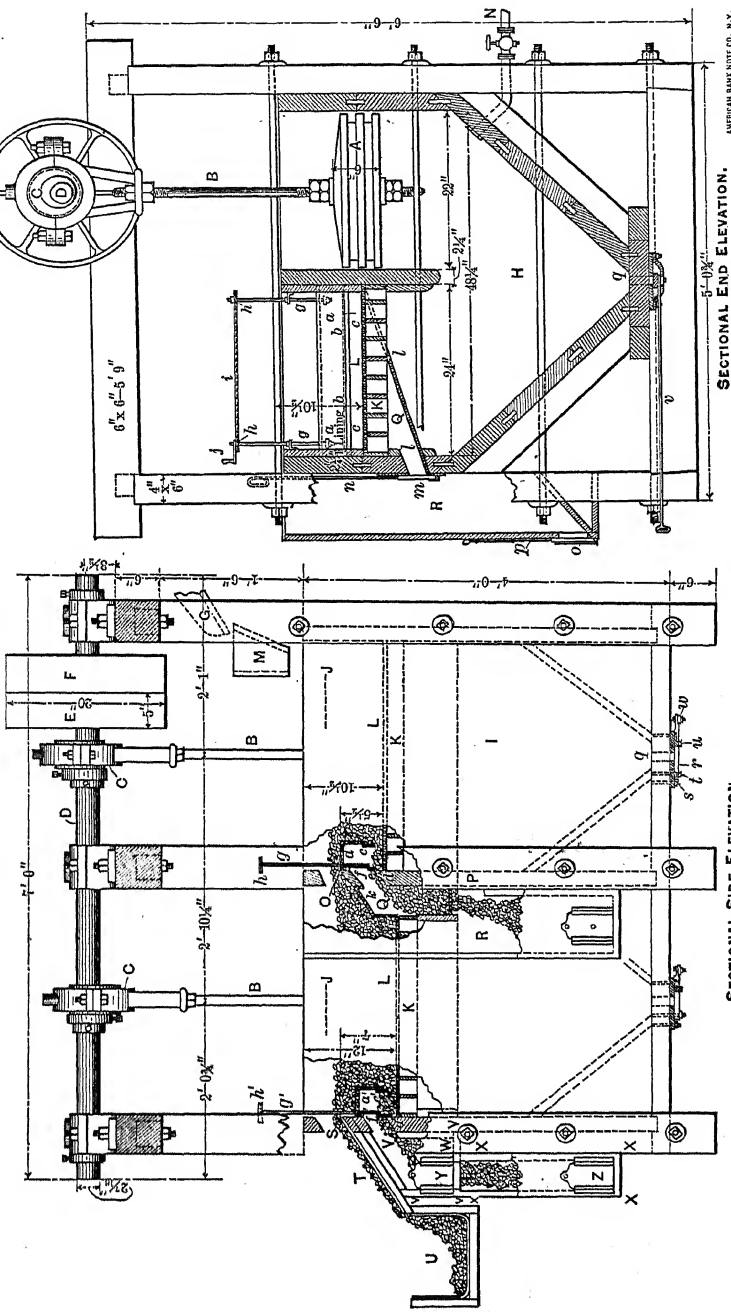
PLAN.

Excentric Jig.

from the receiving- or hopper-end, will have arranged themselves so that upon arriving at the overflows and discharges at the end of each compartment, the particles of less density will be found, to a greater or less extent, nearer the top, while the denser particles will form the bottom layer of the bed, or the lower product, which is then in readiness to be drawn off at the lower discharge as follows:

The material, after travelling the length of the first compartment, I, arrives at the discharge-box, *a*, extending across the width of the jig-bed. The top of this box is about level with the height of the overflow, O, to the next compartment, which, in this case, is $5\frac{1}{2}$ inches above the jig-sieve, L.

FIG. 2.



Excentric Jig.

This box is made of perforated sheet-iron, so that the pulsating and suction-action of the water can be transmitted through it to the material in passing towards and over the upper-product overflow, O. Thus the jigging-action is not interrupted on any part of the jig-frame.

The bottom of this discharge-box, *b*, is at a distance above the jig-screen, which should be a little greater than the size of the particles treated.

The height of the lower product under-flow, *b*, above the screen can be regulated by the under-flow gate, *c*, and will always be a little less than the height of the lower product overflow-dam, *d*, above the screen, which latter can be regulated to more or less than 2 inches by the adjustable dam, *e*. This height will also depend upon the thickness of the lower-product bed which it may be necessary to maintain on the jig-sieve.

The lower product entering the discharge-box, *a*, under the gate, *c*, may then be discharged continuously over the dam, *e*, or intermittently by opening and closing the overflow-gate, *f*, which is made of sheet-iron or grating and opened by being screwed down by the rods, *g*, which are moved by the pinion-wheels, *h*, connected with the chain, *i*, and operated by the hand-brake, *j*, so that the gate is moved uniformly up and down at both ends.

The lower product flows over the dam, *e*, and passes through an opening made in the partition, P, between the compartments of the jig, and of the width of the jig-bed.

The material falls into the lower-product accumulating-box, Q, the bottom of which slopes towards the outer side of the jig. The length of this box extends across the width of the jig-bed, and its width occupies about 4 or 5 inches of the length of the compartment adjoining the one from which the lower product is drawn off.

The box extends below the top of the jig-screen, which is cut away for its insertion; but the jigging action to the material passing over it from the previous compartment is not interrupted in the space it occupies, as the top of the box, *k*, as well as the lower part of the box, *l*, are made of perforated metal, for the transmission of the water-currents.

The lower product can thus be allowed to accumulate in the box, Q, if it does not collect too fast, and discharged intermit-

tently by the gate, m, operated by the rod, n, into the storage-box, R, which is water-tight, extends above the level of the water in the jig and is open at the top.

If the lower product collects too rapidly the overflow-gate, f, and gate, m, can be kept open, and the material allowed to discharge continuously into the storage-box, R. From this box the lower product is discharged by the water-tight gate, o, raised by the rod, p, into the sluice-boxes or other receptacle and conveyed to the point desired.

This arrangement of the lower-product discharge, its underflow and overflow gates and dams, and accumulating-box can be introduced as many times as there are compartments to the jig, the material from each discharging into a separate storage-box, R, or the material from each compartment can all be discharged into one storage-box.

At the final compartment the drawing-off of the lower product is accomplished somewhat differently.

A discharge-box, a' a', similar to that already described, is arranged, as shown, a little back of and below the final point of discharge, where the upper product passes from the jig by the overflow, S, over the apron-chute, T, to the sluice-box, U, or other point of delivery.

The passage of the material into the discharge-box, a' a', and the arrangement of its parts, b', c', d', e', f', g', h', i', j', are similar to those already described.

The material after passing over the overflow, d, falls into a box, V, which is outside of the jig-bed, and not subject to the jigging-action of the water as in the former case, where the material overflows into the adjoining compartment of the jig.

The bottom, W, of the box, V, slopes similarly to the box, Q, although it is of different dimensions. The box, V, is below the overflow-apron, T, of the upper product, and connects with the boxes, X, X, X, X. These are made water-tight, and outside of the area occupied by the apron, T, they extend above the water-level of the jig and are open at the top.

The lower product discharging into the box, V, is discharged by the water-tight gate, Y, into the storage-box, Z, from which it is tapped from time to time, as desired, into a sluice-box, or otherwise conveyed to the desired point.

The perforated metal of the discharge-boxes has holes of

about the same mesh as the screen, L, of the jig-bed, or somewhat smaller in size than the size of the material treated.

The material resulting from abrasion of the particles rubbing together on the jig-bed, and from other sources, which is too small to be maintained on the jig-screen, falls through to the bottom of the box and is discharged by a valve at the hole, *q*. The valve consists of a flat piece, *r*, making a close rubbing surface with the plate, *s*, hinged at *t*, and sliding in a bearing, *u*. The rod, *v*, is connected to the end, *w*, of the valve, and by pulling or pushing the rod the valve is made to close or open the hole, *q*, thus checking or allowing the escape of the fines collected in the bottom of the box.

Middle-Product Jig.

With Adjustable and Automatic Discharges for the Middle and Lower Products.

BY EDGAR G. TUTTLE, NEWARK, N. J.

(Colorado Meeting, September, 1896.)

The accompanying figures show a jig arranged for separating the middle product or middlings obtained in the concentration of certain ores, minerals, coal, etc.

In the preparation for sizing, prior to jigging, of ore-bearing minerals, coal, etc., with their impurities, it is generally desirable to avoid reducing them to unnecessarily small sizes. Where the impurities adhere to the mineral particles or are disseminated through its mass, it is generally necessary to crush to such a fineness as will unlock the minerals from the gangue.

If a considerable proportion of the material consists of mixtures of the minerals and gangue, each in large size, and a lesser proportion consists of a mixture of the two, in smaller sizes of various dimensions, it is not always desirable to reduce the whole to such size as would be required to liberate completely the pieces of ore and gangue in the finest mixtures. In such a case it is preferable to reduce all the material to such an

average size as will completely unlock from the gangue a larger percentage of grains of ore, leaving a proportion of grains composed of part ore and part gangue, not too large to be readily treated and obtained as a middle product from the jig. When the proportion of such middlings becomes too large, a somewhat finer reduction of the material at the start is required.

An output reduced as above and then separated in sizes is ready for treatment, each size separately, on a middle-product jig. What percentage of middle product will result in the various sizes, from the largest down, will depend considerably upon the material and the extent of reduction and sizing.

The construction and operation of the jig used for this purpose is similar to that of other jigs with the exception of the arrangement of the discharging at overflows and gates and the method of gathering the products.

The jig consists of one compartment, in one part of which is the jig-frame, K, and screen, L, and in the other part the plunger, A, which is operated through the rods, B, stems, C, and shaft, D. To one end of this shaft is attached the slide-yoke or crank-arm, E. This is operated by the driven-pulley, F, through the shaft, G, and crank-disk, H, on which is the pin, I, fitted into brasses sliding in the slide-yoke. With each revolution of the crank-disk the arm of the slide-yoke is moved slowly upward and then rapidly downward, thus producing a quick down-stroke and a long, slow return of the plunger.

This motion is best for jigging mixed sizes. For closely-sized products the excentric stroke can be used, or such other as may be required to suit the circumstances.

The jig is filled with water to the level, J J, the supply of water being kept up from the faucet, N. The material to be jigged is fed through the hopper, M.

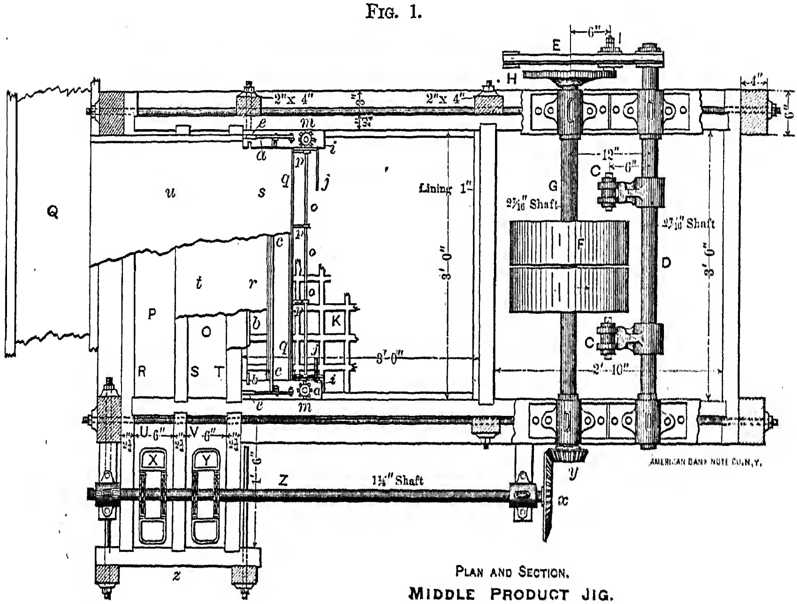
With each down-stroke of the plunger the short, quick pulsion of the water is communicated to the material on the jig-bed, causing the specifically lighter particles to rise higher than the specifically heavier of the same size, and with each return motion of the plunger the particles are allowed to settle slowly, the heavier one falling more rapidly towards the bottom, while the lighter ones, not falling so fast, remain nearer the top.

Thus the particles are treated in travelling across the jig-bed

from the receiving- or hopper-end, and by the time they have arrived at the opposite or discharge-end a separation has been effected, and the mineral of less density is found in the top layer, while the denser mineral forms the bottom layer of the bed.

Particles consisting of a mixture of ore and gangue, and thus having an intermediate specific gravity, will settle as middlings in a horizon between the upper layer of the lighter mineral and the lower layer of the heavier mineral. This product is

FIG. 1.



PLAN AND SECTION.
MIDDLE PRODUCT JIG.
Scale 1 in. = 2 feet.

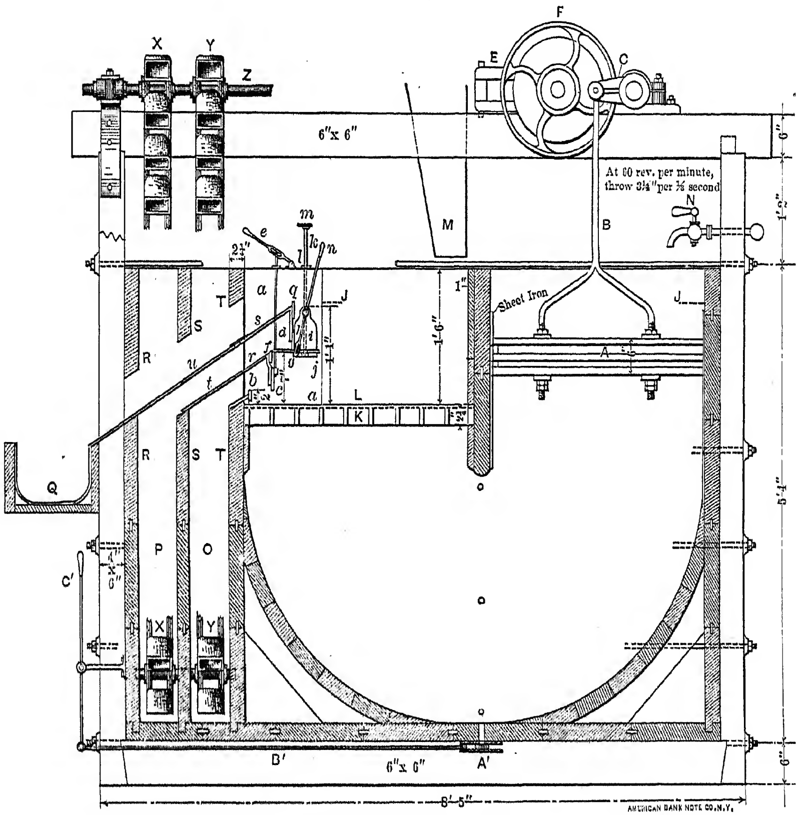
then in position to be drawn off, at the middle product discharge, as follows:

The frame, *a a*, extending across the jig-bed, forms the support for the various dams, gates and overflows for each product which extend across the full width of the jig, so as to avoid the small, narrow, and frequently ineffective, discharges generally used in small jigs. This frame fits closely in position within the jig-lining and is readily removed for repairs or adjustment.

The lower layer of material on the jig-bed flows over the dam, *b b*, into the compartment, O.

This dam extends across the full width of the jig, and, being adjustable, can be raised or lowered, so as to make the height of the top of the overflow more or less than 2 inches. A gate, *c*, extending across the width of the jig, can be closed down onto the jig by the rods, *d*, and lever, *e*. This gate may be kept

FIG. 2.



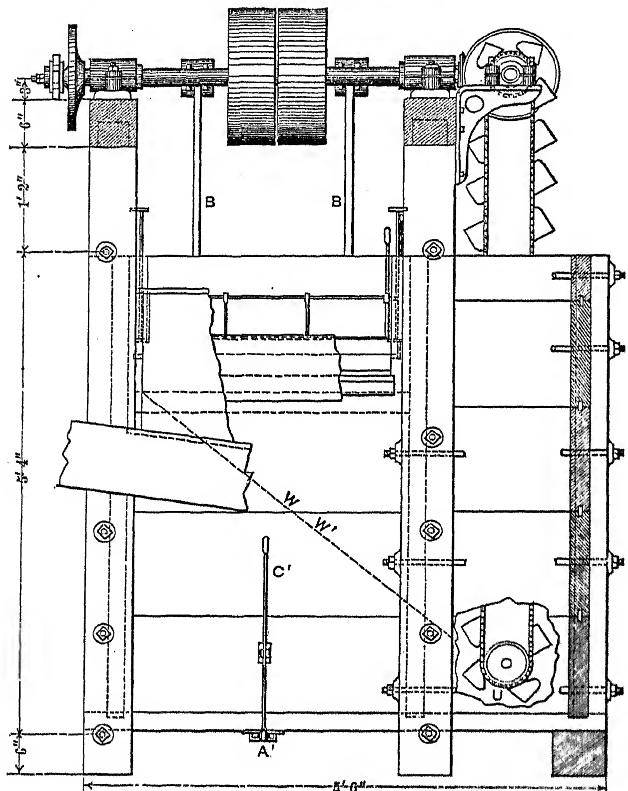
SIDE ELEVATION AND SECTION.
MIDDLE PRODUCT JIG.
Scale 1 in. = 2 feet.

closed and opened intermittently for the discharge; or it may be left open if a continuous discharge is desired, which may have to be regulated with a greater or less opening, depending upon the rate of accumulation of the material of the lower horizon, and the amount of bed of this material maintained on the jig.

The height of the middle-product overflow, *f*, will depend upon the height of this lower bed. With any increase in the height of the overflow, it is necessary to raise the middle-product gate, *g*. This is accomplished as follows:

A piece of sheet-iron, *h*, forming the front-support for the

FIG. 3.



END ELEVATION AND SECTION.

MIDDLE PRODUCT JIG.

Scale 1 in.=2 feet.

gate, extends across the width of the jig, and is bent at the two ends to form the wings, *i*. A piece of sheet-iron, *j*, connecting the wings at the rear, forms the back support for the gate.

The gate, *g*, is made with the slots running in the direction of its width, or else made of wires or small-sized flat iron so as to form an open grate through which the water-currents of the jig

are free to operate, but not large enough to let the particles pass through.

These grate-bars are fitted into openings in the front and back plates, *h* and *j*, to form guides for the gate.

To the back of the wings, *i*, lugs are connected at the top and bottom, which fit into a groove in the frame, *a a*. The rods, *k*, extend down through holes in the frame to the lugs, in which they are free to revolve, but are connected to them so that the wings with the middle-product gate and its support can all be raised or lowered by the screwing of the rods, *k*, up or down in the nuts, *l*.

At the top of the rods, *k*, are pinions, *m*, which are connected with a chain across the jig so that the adjustment of the gate and its supports will be the same at both ends.

The middle-product gate, *g*, and overflow dam, *f*, are thus raised the desired extent, which, as already observed, will vary with the size of the material treated and the depth of bed necessary for this product, as well as that necessary for the product of the upper layer; and also with the rapidity with which the material may accumulate.

The middle-product discharge may be made automatic and continuous by leaving the gate open; or if the conditions are such as to require an intermittent discharge, this may be obtained by occasionally opening the gate.

The gate is operated by the lever, *n*, which rotates the shaft, *o*, which in turn throws the arms, *p*, opening and closing the gate.

The material of the upper horizon of the jig-bed overflows the dam, *q*, which is also adjustable and can be raised or lowered as may be necessary, to increase or diminish the height of the bed.

The parts, *r* and *s*, are of sheet-iron and bent so as to form the chutes for the overflowing middle product and the upper product respectively. These are stationary parts of the frame, *a a*, and can be connected with and disconnected from the sheet-iron, *t* and *u*.

The former product is thus conveyed to the compartment, P, and the latter to the trough, Q, whence it is conveyed to the place of disposal.

The parts, *r*, *s*, *t* and *u*, are fastened securely to the jig-lining,

R, S and T, at intervals across the jig-overflows, so as to stand the pressure from the force of the water which fills the jig and compartments, U and V, up to the level of the upper product dam, and also the compartments O and P, up to the sheet-iron chutes, *t*, *r*, *u* and *s*.

The middle product, after passing out of the jig-bed and over the dam, *f*, slides over the sheet-iron, *r* and *t*, into the compartment, P, where it falls on a chute, W, and passes to the elevator, X, in the compartment, U.

Similarly, the product at the bottom of the jig-bed passes over the dam, *b*, and falls upon a chute, W′, in the compartment, O, whence it passes to the elevator, Y, in the compartment, V.

The elevators, X and Y, are fitted with perforated buckets and lift the two products from the compartments, U and V, above the level of the water, J J. The material is thus drained and discharged at the elevator-heads.

The elevator-buckets are fitted on a sprocket-chain, operated by 6-inch sprocket-wheels on the shaft, Z, at one end of which is a bevel-gear, *x*, 12 inches in diameter, driven by a 6-inch bevel-gear, *y*, fitted on the shaft, G.

The side of the elevator-compartment, *z*, is bolted to the frame-work of the jig and is readily removable for access to the foot of the elevators in case of repairs.

The fines falling through the jig-screen are drawn off intermittently by the gate-valve at A′, which is opened and closed by the lever, C′, through the rod, B′.

The jig here shown is of wood. The arrangement for drawing off the different products can be equally well-fitted to a jig made of iron. The jig-bed and plunger are each 3 feet square. These may vary in dimensions and may be equal or unequal in area.

Gold in Granite and Plutonic Rocks.

BY WILLIAM P. BLAKE, TUCSON, ARIZONA.

(Colorado Meeting, September, 1896.)

A RECENT paper by Prof. George P. Merrill, Curator of the Department of Geology of the U. S. National Museum, Wash-

ington, upon "An Occurrence of Free Gold in Granite,"* describes an interesting instance of the dissemination of this noble metal in the substance of granite of normal composition believed to be from Sonora, Mexico. He found the gold in small scales, rarely exceeding a millimeter in diameter, distributed through the scales of mica and apparently enclosed in both the feldspar and quartz granules. A number of thin sections of the rock submitted to examination with the aid of the microscope gave confirmatory evidence. Sulphides were not detected, nor any indication of a secondary impregnation, though the granite had undergone some alteration, apparently by weathering. Mr. Merrill concludes that "there is apparently no way of accounting for the gold other than by considering it an original constituent of the rock, a product of cooling and crystallization from the original magma."

We thus have another link in the chain of evidence showing that gold is a constituent of granite and of plutonic rocks, and that such crystalline rocks may be the primal source of the gold, which is concentrated in veins.

We do not, however, overlook the fact that the oceans may have contributed a portion of their dissolved gold to the sedimentary rocks, such as the slates, magnesian or otherwise, ot different geological epochs; such slates being generally known to us as the country, or wall-rocks, of most of the auriferous quartz-veins, especially of the central gold-region of California.

As investigation progresses, and our knowledge is increased, it becomes more and more evident that such pyritous sediments derived their metallic contents from the waters of the ocean at the time of their sedimentation, through the reducing agency of organic matters, or the exhalation of sulphuretted or carburetted gases, as, for example, from the petroleum shales.†

We encounter here the difficulty that gold is not uniformly present in pyritous deposits, as we might expect considering the universality of the oceanic source, and the general distribution of gold in this menstruum. But before we can generalize satisfactorily we require much more evidence regarding the dissemination of gold in the mass of the various kinds of rocks,

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* *American Jour. Science* (iv.), i., 309, April, 1896.

† See my remarks on the "Lead- and Zinc-Deposits of the Mississippi Valley," *Trans.*, xxii., 621, 629 and 630.

independently of veins. Examples of the nature cited by Prof. Merrill are thus of great importance, not only in themselves, as bearing upon the question of the origin and distribution of gold, but as tending to stimulate observation, inquiry and discussion. Prof. Merrill refers to the description by J. B. Jaquet of an occurrence of free-gold in microcline in a rock consisting essentially of microcline and quartz impregnated with hematite,* and, also, to the occurrence of free gold in glass-like and crystalline varieties of a quartz-trachyte, in Chili, S. A., as described by Möricke.†

It is only within a few years that the instances of the occurrence of gold in the crystalline rocks have multiplied until we cannot longer regard them as exceptional, or hold that gold, as a rule, is confined to the magnesian and argillaceous slates.

One of the earliest cited‡ and best examples of the occurrence of coarse free gold in the midst of granite, seemingly without any extraneous origin by impregnation accompanied by quartz, was at the Armagosa gold-mines at the sink of the Mojave river, San Bernardino county, California.

The native gold in visible masses was there disseminated in the midst of the aggregation of soda-feldspar (albite) and the quartz, as if an original constituent of the mass and indigenous to the rock, which it may have been. Pyrites were not present in the specimens obtained; but some fragments of white arsenic in granite from the same locality may be regarded as an indication of the former presence of an arsenide in association, and as possibly an impregnation subsequent to the formation of the rock. The specimens, unfortunately, have been packed for years and are not now accessible. In all cases of this kind it is important to study the rock *in situ* and in mass, and to note carefully all the surrounding conditions. It is not safe to make deductions from the phenomena exhibited by isolated specimens.

The influence of dikes of plutonic rock upon the mineralization of veins with the precious metals, has long been known and recognized by miners.

If we accept the theory of lateral secretion for the formation

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* "Geology of the Broken Hill Lode," etc. *Memoirs Geol. Survey of N. S. Wales, Australia*, No. 5, 1894.

† *Tschermak's Min. u. Petr. Mittheil.*, iii., 1891.

‡ Blake, *California Minerals*, 1864.

of mineral veins, such veins bearing the precious metals are perhaps as good evidence as we need of the diffusion of gold and silver in the mass of the adjoining country-rock, or, at least, that solutions bearing these metals may traverse the rocks by osmosis or otherwise. If such evidence may be admitted, the range of the phenomena and of the evidence is widely extended.

Some of the most notable districts in California where gold-bearing veins traverse crystalline rocks of the granite family are at Grass Valley and Nevada, Nevada county; also at Forbestown, Butte county; Ophir, Placer county; and West Point, Calaveras county. In these localities the veins are numerous, and appear to have been formed from the substance of the country-rock. In Mariposa county, at the southern end of the Mariposas estate, where the slates of the Jura-Trias age, with their accompanying large "Mother-Vein," give place to granite, we find a vein of auriferous quartz which may not have any relation in origin to the vein in the slates, with which the vein in the granite contrasts strongly in its formation and in the distribution of the gold. It is known as a "pocket-vein," yielding the gold in isolated but rich bunches, and of a higher grade of fineness and less crystalline than the gold of the Princeton vein, which traverses the secondary slates. It would be instructive to determine the relative ages of these adjacent veins and the influence of the two different kinds of country-rock upon the deposition of the gold. It would be extremely instructive if we could find an instance of a gold-bearing vein passing from one formation into another, as, for example, from slates into compact granite, so as to exhibit the effects of change of country-rock, or of the walls, upon the mineralization of the vein—effects so strikingly shown in the lodes of Cornwall, which are copper-bearing in the "killas" and tin-bearing in the granite. It should be mentioned that the tin-lodes are not without some gold, being thus indicative of its presence in the granite.

The well-defined granite rock of Butte and Walkerville, Mont.,* affords conspicuous examples of gold-bearing and silver-bearing veins, originating apparently by lateral secretion

* For descriptions, see the papers of Mr. Emmons, *Trans.*, xvi., 49, and of Mr. Brown, *Trans.*, xxiv., 543.

from the body of the granite on each side. The Rainbow* and Blue Bird lodes especially appear to have derived their mineral and metallic contents from the granite in much the same way as the tin-bearing lodes are formed in Cornwall, according to the investigations of Le Neve Foster.

The occurrence of gold in thin flakes upon the surfaces and in crevices of the porphyry at the famous Contention mine at Tombstone, Arizona, has been shown;† but it is not yet known with certainty whether this gold, either in its free state or combined with disseminated pyrite, was an original constituent of the dike, or whether it was derived from the diffused pyrites of the stratified beds traversed by the dike. So far as regards the plainly visible gold, it appears to be confined to the partly decomposed portions of the porphyry dike, at or near the contact with the other rocks, and to bé a secondary or late deposition and not indigenous.

The mines of the Homestake group in the Black Hills of Dakota afford a good example of the occurrence of gold in ancient crystalline gneiss or granitic schists of pre-Cambrian age. These schists are much plicated and are traversed by felsitic dikes, distinctly intrusive. Whether these are auriferous or not remains to be ascertained, and without a careful examination it is not possible to state the source of the gold, whether indigenous to the schists or introduced from the dikes or with quartz-veins from remote sources. The rock for the stamp-mills is quarried, rather than mined in the manner usual with veins, and the rock is milled, together with any veins traversing it.

The ore of the Treadwell mine, Alaska, is described by Adams and Dawson‡ as a hornblende-granite, "much crushed, altered, and impregnated with secondary quartz, calcite, and pyrite." The pyrite contains the gold.

In the gold-bearing region of Northern Sonora, Mexico, the gold-veins are chiefly in or closely associated with granitic and plutonic rocks. The veins of El Grupo concession, about 100 miles southeast of Tucson, Arizona, traverse a fine-grained granite, and bear both gold and silver. A dioritic rock at El

* See the writer's paper on the Rainbow lode, *Trans.*, xvi., 65.

† *Trans.*, x., 340.

‡ *American Geologist*, August, 1889. Cited also by Merrill.

Plomo, in the same State, appears to be especially favorable to the occurrence of gold. At the San Francisco mines, 100 miles southwest of Gila Bend, Arizona, gold-bearing quartz-veins traverse Huronian or pre-Cambrian gneiss near or at the contact with a great dike of diorite on one side and of porphyritic granite on the other. El Campana, a large quartz-vein, also in Sonora, is at the contact between an obscurely defined felsitic rock, like granite, on one side, and a heavily-bedded quartzite (probably Cambrian) on the other.

In Central and Northern Arizona gold-bearing veins are found in granite. The famous Congress vein now worked to a depth of some hundreds of feet, traverses granitic rock in close association with a dike of plutonic rock. North of Indio, on the Colorado desert, California, in the second range of mountains, a ridge of granite contains irregularly spread patches or bunches of pyrite which, by decomposition, liberate a small amount of gold. This pyrite does not appear to be connected with any vein, but seems to be one of the original constituents of the rock. Somewhat similar pyritic impregnations in granite were worked a few years since in Arizona, near Peeples Valley and Rich Hill, north of Stanton, and yielded free gold.

The close association of gold and granite at Cripple Creek must not be overlooked. Some specimens of porphyritic granite with cavernous spaces partially filled with purple or amethystine fluorite are very rich in gold.

Penrose, in summarizing Chapter IV. of his monograph of the mining geology of the Cripple Creek district, says:

"The gold and associated vein-minerals were probably derived from the volcanic rocks and, to a less extent, from the adjacent granite, at greater or less but not extremely profound depths."*

Again (p. 150), "The most profitable mines yet discovered in the Cripple Creek district are in the eruptive rocks or in the granite immediately adjacent to the main volcanic vent or vents."

On Battle mountain, Cripple Creek, the veins are described as in some cases well-defined bodies of quartz, "in other cases they are impregnations and partial replacements of the country-rock with mineral matter along fissures. The prominent veins

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* *U. S. Geol. Survey,* 16th Rept, Pt. ii., 1894–95, p. 166.

are both in the breccia and in the granite immediately adjoining the breccia area."* This breccia, as found at the Independence No. 4 mine, is described as containing, besides volcanic rocks, large quantities of granite fragments, some of them several feet in diameter.

The ore of the Independence mine consists of "granite from which the mica and quartz have been partly or wholly removed, leaving a honey-combed, vesicular mass of partially kaolinized feldspar" (p. 201).

The phenomena of the occurrence of the precious metals in the celebrated Mercur mining district, Utah, furnish evidence which may be used in support of either view of the primal source of these metals. Both sedimentary and plutonic rocks are there found in conformable layers, and gold has been found in both, but the commercially available deposits are along and near the plane of contact of porphyry and limestone, but below the porphyry, the deposits thus being what is ordinarily described by miners as a contact deposit. According to the researches of Mr. J. Edward Spurr, there is a slight mineralization which is pretty generally distributed throughout the rocks of the basin. "Four assays were made of comparatively fresh Eagle Hill porphyry. Two assays showed 0.01 ounce of gold to the ton; one showed a trace of gold, and the fourth was entirely barren. Of two assays of the birdseye porphyry, one showed 0.01 ounce of gold to the ton, the other showed a trace of gold."†

Nine assays were made of the altered limestone, seven of which showed very small quantities of gold, and two did not yield a trace. Of twelve assays of rock at the contact where there was no evidence of mineralization, nine yielded small quantities of gold, and the other three did not show the presence of the metal.

The phenomena indicate to me that the porphyries and possibly the shales above them are the sources of the gold, but Mr. Spurr concludes that the thickness of the porphyry-sheet is so slight that it is not possible that the ores could have been derived from it by leaching, and so carried downward into the

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* Penrose, *Ibid.*, p. 200.

† "Economic Geology of the Mercur Mining District, Utah." 16th Report *U. S. Geol. Survey*, Part ii., 1894–95, p. 381.

limestone, and his explanation, or theory, is that "the mineralizing agents rose from below till they met the sheet of altered porphyry, when they spread out along the under contact and so produced the mineralization."*

He also found a series of nearly vertical fissures or fractured zones through which he believes communication was established with a body of uncooled igneous rock at an uncertain depth below, permitting the ascent of moist volcanic vapors (p. 453).

The researches of Emmons upon the dissemination of gold in the rocks at Leadville have added largely to our knowledge of this subject.

In the representative series of gold specimens and ores sent by the Minister of Mines and Agriculture from New South Wales to the Columbian Exposition in Chicago in 1893, there were numerous specimens from granitic and feldspathic lode-stuff. Nos. 10, 11 and 12 of the collection were from "binary granite." No. 10 contained gold associated with copper pyrites and iron pyrites from the Challenger mine, Adelong. No. 11 was from the 978-foot level of the Great Victoria mine, and No. 12 from the 770-foot level of the same mine.

A pyritous granite was shown from Dargue's Reef, Major's Creek, Braidwood. This auriferous stuff is described as 25 feet in width at the 225-foot level. Nos. 23, 24, 25 and 27 of the same collection consisted of auriferous feldspathic lode-stuff from different mines at Yalwal, some specimens showing free gold. Feldspathic lode-stuff with mispickel and oxidized pyrite is found also at the Junction Reef, Mandurama. Auriferous quartz and feldspathic vein-stone rich in gold occur in the Hill End district and at Delaney's Dike, near Molong.

In the collection from Sydney there were gold-bearing specimens with feldspathic gangue from Saw Pit Gulley, Fairfield. Three mines at Timbarra were represented by masses of auriferous granite. The feldspathic gangue occurs in some places in a brecciated condition. At Wann's lode, Drake, New England (No. 140) the gold is obtained from a siliceous feldspathic breccia. At the Mount Graham gold-mine, Pambula, the occurrence of the gold is described as "unlike anything hitherto discovered in any of the Colonies. The lodes are in the main

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* *Ibid*, p. 449.

conglomerates and felsitic breccias, in many instances only to be distinguished from the country-rock by irregular walls. The gold is extremely fine and difficult to follow; frequently there is nothing to distinguish the gold-bearing from the barren portions; the drillings and the mortar are the only guides."

Specimens of vesicular and amygdaloidal basalt, claimed to be auriferous, were shown from Black Rock, Bullina.

A careful chemical investigation for gold of the basalt of Ovifak, Greenland, which contains the large and small masses of metallic iron, and which is believed to come from great depths in the earth's crust, would be exceedingly interesting.

Examples of the presence of gold in granitic and plutonic rocks might be multiplied, but those given are sufficient to show that we must recognize such rocks as truly gold-bearing.

Rapid Section-Work in Horizontal Rocks.

BY MARIUS R. CAMPBELL, U. S. GEOL. SURVEY, WASHINGTON, D. C.

(Colorado Meeting, September, 1896.)

EVERY mining engineer who has engaged in prospecting for coal in flat-lying rocks understands the importance of constructing geological sections across the territory which he has to prospect. If the area is small and the time and means at his command are unlimited, he can construct such sections with the transit and level; but if the area is great and the examination hurried, as it frequently is, he will doubtless be puzzled how to proceed. Under this pressure of time and means, his work will probably be, to a great extent, devoid of any system, being simply a hap-hazard opening of such seams as show in outcrop at some point on the territory, without any determination of geologic structure or stratigraphic succession.

Prospecting carried on in this manner is never satisfactory to the intelligent mining engineer, for he must realize that the information which he is able to furnish to his employer is totally inadequate to determine the economic value of the property. To the owner of the land such a report is no more satisfactory than to the engineer. Unless the latter can show both

the stratigraphic relations of the coal-seams and the geologic structure, the owners are dependent entirely upon his statement concerning them, and have no means of judging for themselves as to the accuracy of his work, or the value of his conclusions. Even supposing that the report is accurate so far as the number and thickness of the seams and the quality of the coal are concerned, the owners are still very much at sea on the question of the location of mines and the direction in which the work should be carried under the surface. Only one class of men might profit by an unscientific report, namely, those who are seeking to unload worthless property on an unsuspecting public. They succeed best when they are unhampered by facts and figures; but to the *bona fide* operator who seeks information upon which to base his plans for development, such a report can be of but little assistance.

The geologists of the United States Geological Survey, whose work has extended into the great Appalachian coal-field, have felt this need perhaps more than the mining engineer; for the work of the geologists is necessarily rapid, lacks the facilities for thorough prospecting, and involves territory of enormous extent. Under such conditions the ordinary methods of conducting geologic research failed signally, and a new method had to be devised, if the work was to go on.

The writer, having been instrumental in introducing, for the conduct of such work, a new system, which has met with general approval wherever it has been tried, deems it a fitting subject to present to the Institute.

The Problem.

Throughout the middle and southern portions of the Appalachian coal-field the strata are monotonous in character, consisting of shales, coals and sandstones. Limestones are almost unknown; and, in the regions where they do occur, they are irregular in their distribution, and therefore of but little value for purposes of correlation or the determination of stratigraphic succession. Conglomerates occur frequently, but from their very nature it is hardly probable that any one bed extends indefinitely in any direction. Considerable experience has shown that it is not safe to assume that a bed of conglomerate which is heavy in one place will be found in an adjoining locality. It

is also rare that a sandstone is found to have sufficient individuality to be recognizable over wide areas. Shales are, as a general rule, totally unreliable as guides, from the difficulty of recognizing slight peculiarities which any one bed may possess, and also their liability to change in character from point to point. Coals are, perhaps, the least reliable means of correlation. True, they have been generally relied upon; but the glaring errors which have been made in correlating seams is sufficient evidence that, taken alone, the coals are of doubtful and uncertain value in carrying correlations from point to point in one field, or from one field to another. Since no one stratum can be depended upon implicitly, our method of work must be such that each stratum can be traced wherever possible, and correlations can be made from the aggregate evidence of all of the beds, whatever their character may be.

In order to do this, all the evidence collected in the field must be placed upon sections in such a graphic manner that the geologist can see at a glance the material at his command, and can give to each fact its proper value in making his correlations. This implies the construction of sections in the field; and to satisfy the conditions, they must be rapidly and easily made, without instruments other than those which can be carried in the pocket, and the sections must be so arranged that they can be made by the geologist when he travels on foot or horse-back, or drives in his cart or buck-board.

The rocks over most of the field are essentially horizontal, but when considered in detail they are found to be very irregular. The general geologic structure is necessarily simple, and yet the minor irregularities are so numerous and apparently so devoid of system that the question becomes at times very complex.

Solution of the Problem.

Since the rocks are nearly horizontal, the vertical element in the section is the most important. Time will not permit the use of the Y-level, consequently we are limited to the aneroid barometer and hand-level. The aneroid barometer is generally regarded as unsatisfactory; but in order to do rapid work the use of this instrument is a necessity. The hand-level should be used to supplement the aneroid on steep slopes and where horizons are well marked.

The note-book should be as large as the geologist can conveniently carry. The book in use in the United States Geological Survey is $7\frac{1}{2}$ by 10 inches in size, and made of cross-section paper. The size of the book is not important; each engineer must determine that question for himself.

The method here presented presupposes a good base-map of the region to be studied, upon which are represented all of the roads and, if in an uninhabited or but thinly settled region, all of the more important bridle-paths. If such a map cannot be obtained, the geologist will perforce prepare his own map at the same time that he constructs his section. Engineers accustomed to mapping in a thoroughly accurate manner will find it difficult to adapt themselves to simpler methods, which, though not strictly accurate, are correct enough for geological purposes. Through long experience, the geologists of the United States Geological Survey are able with a hand-compass and a protractor to make foot-meanders with considerable accuracy and great rapidity. Space will not permit a full discussion here of methods of rapid mapping; but it will repay engineers to practice it in regions where accurate surveys are impracticable.

In order to render the explanation of the method of making rapid sections more intelligible, we will consider a concrete case taken from my own note-book and representing a section made in this manner in McDowell county, West Virginia. Plate I. shows a plat of the road over which the section was made. This road follows a high ridge on the western side of Dry fork of Tug river, and is the principal line of communication between Bradshaw and Iaeger. For this kind of work, ridge-roads generally give better results than those following streams, since on the summits of the ridges sandstones and shales are almost always distinguishable. The section herewith presented was made during one trip over this road, and consumed less than half of a day for its completion. The only instrument used was an aneroid barometer, the readings of which were afterwards checked by known elevations along the line of the section.

Since the method is almost entirely graphic, it is well to adopt conventional symbols to represent the various kinds of rocks included in the section. Of course, all this is a matter of personal taste; but much time is saved by such symbols as

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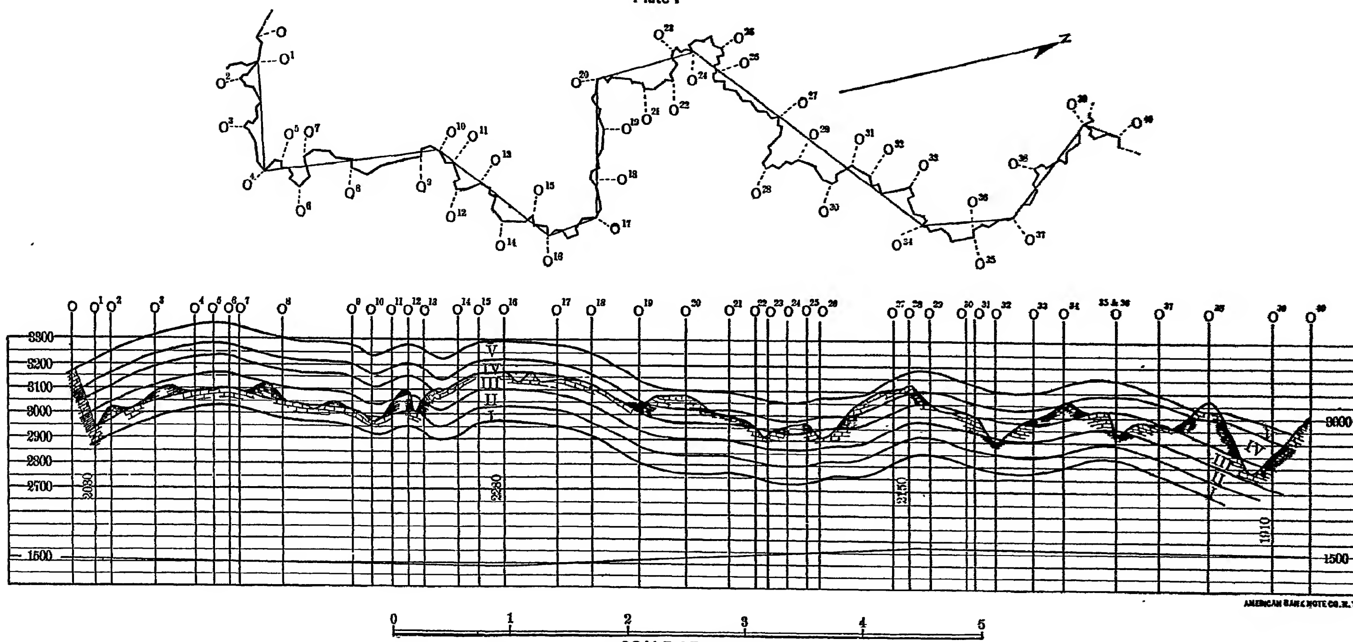
Plate I



SCALE OF MILES

Plat and Section in McDowell county, W. Va.

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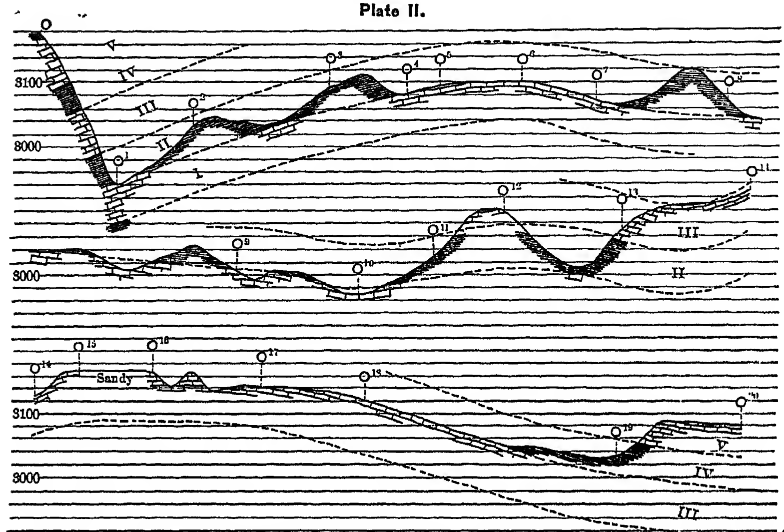
are easily made and easily recognized afterward. In the section illustrated I used a block-symbol for sandstones (varying the size of the pattern as the sandstone changed in the thickness of its bed), fine-line shading for shales and a heavy black line for coal-outcrops. The number and kind of symbols which it is desirable to use will be determined by the character of the rocks involved and the ingenuity of the person making the section.

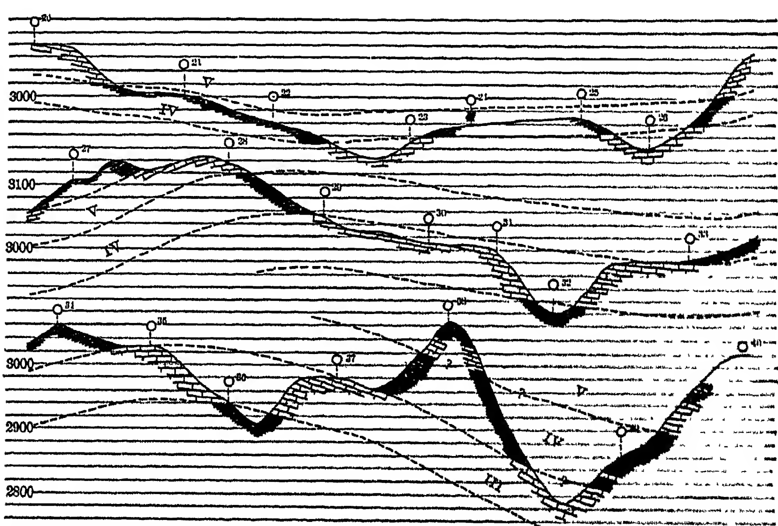
Plate II. is a reproduction of the pages of my note-book upon which the original section was made, the notes having been inked for the purpose of mechanical reproduction. The note-book was carried open in my hands, and the barometer was held upon the open book, so that I could always read the elevation of any outcrop without the necessity of stopping the team for the purpose.

In constructing the section the horizontal element is almost ignored; but the vertical is made as nearly exact as possible under the circumstances. A convenient vertical scale is adopted, which should be large enough to show all the details of the section. The scale used in the example here given is 20 feet to a single space. In the original book this is about 100 feet $= \frac{5}{6}$ of an inch; but the scale may be varied to suit the book, the character of the rocks and the inclination of the person doing the work. Every observed outcrop is plotted in its correct relative position, as far as the vertical element is concerned; but the horizontal distances are merely estimated —the hills and valleys being sketched as the section is made up, without any particular effort to have them correct, except in elevation. At the starting-point, which was marked O on both map and section, the barometer read 3180 feet, which was assumed to be the elevation without any regard to the absolute elevation. In passing down the road the shale which caps the hill at the starting-point was found to rest upon a bed of sandstone at 3170. This sandstone is coarse and heavy-bedded, and reaches to 3100. The contact between the shales and sandstone was seen at this point, as was also the contact between the shales and a lower sandstone at 3060. This is also a coarse sandstone and extends to 2990. Below that is sandy shale to the foot of the hill, where the intersection of two roads provides us with a tie-point, so our section is marked O^1,

Plate II.



ROAD FROM BRADSHAW TO IAEGER.



ROAD FROM BRADSHAW TO IAEGER.

AMERICAN BANK NOTE CO. N.Y.

Note-Book Sketches.

and the road-intersection on the map is also marked the same. This point is on the top of another bed of sandstone, which is well exposed on the road which descends from this point to Bradshaw. From this side-section we know that the sandstone is 65 feet thick; so it is plotted in on our section to assist in the correlation, if the road should cut this bed at any other point on the section. For convenience in describing the construction of the section, the various beds are numbered, beginning with the lowest sandstone shown in the section. Thus the sandstone beds are called Nos. I., III. and V., and the shale-beds Nos. II. and IV.

Many written descriptions of the kinds of shale and sandstone were originally made on this section; but it was impossible to reproduce all, so they have been generally omitted.

In passing eastward from the point O^1, the road ascends on the shale for a vertical distance of 90 feet to a sharp bend in the road, which is used as another tie-point and marked on the plot and section O^2. The attitude of sandstone No. I. below the road is unknown, until the road passes over a low hill and descends into a shallow saddle, where the sandstone is seen immediately underlying the road. Since shale bed No. II. is exposed continuously from O^1 there can be no doubt about the identification of the sandstone stratum.

Throughout the entire section tie-points were established at frequent intervals or wherever a point in the road could be identified.

The road continues in shale bed No. II. past O^3, and at O^4 cuts through the bed of shale and reaches the sandstone below. The road follows near the top of this bed through O^5, O^6 and O^7; but at the last point again arises over a hill composed of shale bed No. II. There is an apparent arch in the sandstone (No. I.) between O^1 and O^7; but upon consulting the map it is found that the axis of this supposed arch is located at the angle in the road which unites the east and west with the north and south courses. There is a descent in each direction from this angle; consequently we conclude that there is a strong northwesterly dip in this region, instead of an anticlinal fold.

The hill between O^7 and O^8 affords a minimum measure of shale No. II.; but since no sandstone was noted upon its summit, the exact upper limit is in some doubt. Beyond O^{10} the road

ascends a hill 130 feet in height. Leaving sandstone No. I. at its base, it passes through about 70 feet of shale, and reaches the basal portion of an upper sandstone (No. III.).

This portion of the section illustrates well the trouble frequently encountered in constructing sections in Coal-Measure rocks. In ascending the hill from O^{10} the road rises distinctly above the sandstone (No. I.) and enters the shales above. These shales show in outcrop through a vertical interval of 70 feet, above which no rocks are visible. The soil is the only means of determining the character of the rocks; but the soil is sandy, and it is impossible to say whether it is derived from the sandstone which caps the hill or is the result of the disintegration of sandy shale or sandstone in the place where the soil occurs. Generally a single exposure will not suffice to settle the point, and the geologist must seek for further exposures of this interval in order to become better acquainted with the character of its rocks. In the section under consideration, this interval was afterward seen in a number of places, and the shale bed (No. II.) was found to be very constant and about 70 feet in thickness. Therefore, it seems probable that the blank space in the section below O^{12} is really sandstone belonging to No. III.

Sandstone No. I. passes below the road at O^{10} and was not seen again in the section.

The section was carried on as already indicated, estimating all horizontal distances and plotting all vertical measures from the reading of the barometer. Cross-references were established at frequent intervals, by means of which these points could be identified in the future, and the section could be reconstructed on a correct horizontal basis.

The various beds were followed without any great difficulty to O^{37}, at which point the road passes upward through the sandstone No. III. and a greater thickness of the shales of No. IV. than had previously been found. The sandstone (No. V.) above these shales seems abnormally thin in this hill, but on the opposite side of the ravine, near the end of the section, it is nearer its normal thickness and is accompanied by a band of thin, flaggy sandstones above the heavy portion. This upper portion may be present on the hill at O^{38}, but it was not noticed.

Whether these changes are due to actual increase and de-

crease in the thickness of the beds or whether they are due to fluctuations of the barometer is uncertain. I had not the opportunity of traversing the section a second time, so could not settle this question.

This practically completes the field-work on this section. Since the horizontal element of the section is merely estimated, it is not to be supposed that the section as constructed in the field will show correct geologic structure, but it does show the correlation of the beds as well as though the section was correct in both its horizontal and vertical elements. The identification of the beds is all that is essential in the field; the structure can be determined later, when the geologist has more leisure and better facilities for the work. True, the section here given is an exceptional one; but in all cases, if exposures are reasonably good, the beds can be followed with considerable certainty. Where exposures are poor, or no rocks at all show on the surface, all methods, however good in themselves, fail to reveal either the geologic structure or the stratigraphic succession.

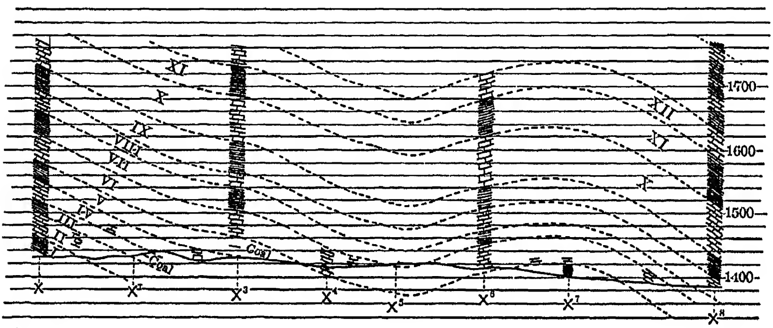
The method of procedure which is applicable to ridge-sections will not always apply to sections along streams. In the latter case, if the observations are confined to the rocks exposed in the bed of the stream, the possibilities are that the section will have but little value. Rock-surfaces, when fresh, are very different in appearance from the same surfaces when deeply weathered; consequently the interpretation of creek-sections is liable to be very different from the interpretation of ridge-sections. Moreover, along streams the shales and softer beds are liable to be worn away by mechanical action, leaving the sandstones as the most prominent, if not the only visible, geologic feature along the stream-valley. Good sections can usually be made along streams, if there are hills bordering the valley, up which the observer can climb to rocks exposed on their slopes. Manifestly the best results can be obtained where the hills are very abrupt and composed of well-defined beds of shale and sandstone, or other material equally well differentiated.

Plate III. is a photographic reproduction of two pages of a note-book, showing a section along Tug river in McDowell county, West Virginia. It would have been impossible to interpret the geologic structure from the exposures along the bed

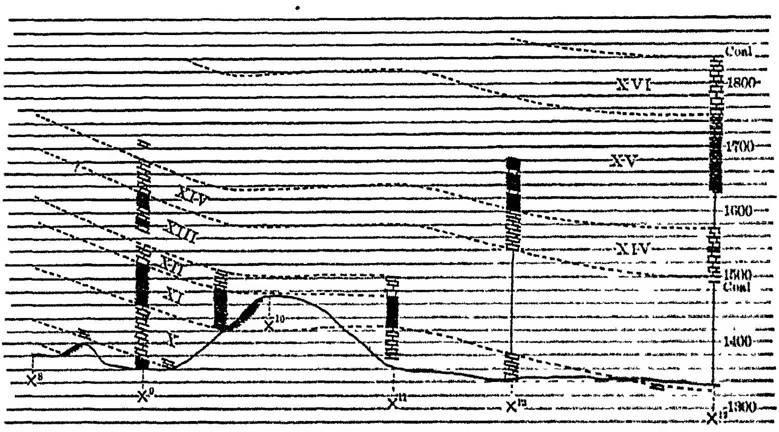
of the river; consequently it was necessary to make frequent sections up the hills bordering the stream.

The start was made from the point X at the the mouth of

Plate III



SECTION ALONG TUG RIVER FROM SAND LICK CREEK TO WELCH



Note-Book Sketches.

Sand Lick creek. A climb was made to the top of the hill, 340 feet above the stream, in which five beds of sandstone were located, and a good determination was made of the thickness of the shale-intervals separating them. The sandstone beds are

numbered on the section II., IV., VI., VIII. and X. and the shale-beds, I., III., V., VII. and IX. On passing down the river from the point X, sandstone No. II. was seen to pass beneath water-level, and the coal which shows in No. III. at X is brought to the level of the stream near X^1, where it has been opened and faced up. Sandstone No. IV. is also seen to pass below the road, and shale No. V., which carries the Pocahontas coal-seam, is also brought down to the level of the road. This coal-seam has been opened in a number of places in this vicinity, the latest opening being at the point X^3. Above this opening a second climb was made, and another good section of the various beds occurring in an interval of 330 feet was obtained. Sandstones Nos. VI., VIII. and X. were found to have the same thicknesses and the same intervals separating them as in the previous section. Sandstone No. X. is especially prominent, and serves as a guide in the identification of formations. In this section the whole of the shale interval above No. X. and a small portion of an upper sandstone were obtained.

In passing down the river from X^3 the coal first disappears, and sandstone No. VI. also passes below water-level, but rises soon and shows prominently at X^6. The disappearance of this sandstone below the level of the river marks the axis of a small synclinal fold.

Fearing lest there might be some error in tracing along the road, another climb was made at X^6. Happily the succession of beds was found to be perfect, and fully verified the tracing along the stream. No. X. is still the most prominent bed and the guide to the stratigraphy. In this climb the full thickness of the upper sandstone (No. XII.) was obtained; for shale was found capping the hill above the cliff formed by this bed of sandstone.

Below X^6 the strata continue to rise to X^7, at which point 20 feet of the shales of No. V. are visible. It is also reported that the Pocahontas coal has been seen here at low water. Beyond X^7 the strata dip so rapidly that another climb was necessary in order to verify the apparent structure. This climb reveals the fact that No. VIII. is at water-level, and that the heavy cliff next above is formed by No. X. No. XII. is also found with a normal thickness of 40 feet. Above No. XII., for a distance of 120 feet to the top of the hill, is an interval in which

there is no bed with well-marked characteristics. The bed of sandstone near the top of the section is inconspicuous here, but apparently swells to a prominent bed farther down the river.

Since this section was unsatisfactory, another was made a short distance below, in which No. X. was obtained near the level of the road. No. XIV., the highest bed in this section, is quite heavy here; so we are forced to the conclusion that there has been a rapid thickening of the sandstones, or some of the sandy shale in section X^8 has been replaced by, or changed to, sandstone in X^9.

From X^9, the road climbs over a projecting spur, and in so doing crosses No. X. and No. XI. It then descends to the river bank, with No. X. forming a rugged wall just above the road. To verify this a climb was made at X^{12}, starting at No. X., and continuing through No. XIV., which is here a prominent ledge, and for a distance of 80 feet into the sandy shale of No. XV.

No. X. passes gradually below water-level, and at Welch, which is located at the end of the section, there is no stratum near the level of the stream by which we can follow the structure. The climb at this point, however, seems to give No. XIV. in increased thickness and introduces a new member, No. XVI., into the section. This sandstone is the most conspicuous feature of the lower river, but unfortunately has been eroded from the cliffs farther up the stream.

Reference-letters are used in the same manner as in the section previously described. If the work is hurried, the vertical sections must of necessity be measured by the aneroid barometer; but if time will permit, a Locke hand-level can be used to great advantage in measuring sections where exposures are good and slopes are steep. The results are much more satisfactory than if all of the measurements are barometric; and, with practice, measuring with the hand-level can be done quite rapidly.

As before mentioned, these sections should be supplemented by such written notes as are necessary to convey to any person a knowledge of the character of the rocks observed, and the details of the coal-seams. If the notes are taken in this manner, a single section will frequently furnish considerable information regarding the geologic structure; but in undertaking

to map a given area, numerous sections should be constructed, showing the details of structure along lines in various directions across the territory.

Office-Work.

The rough sketch of the field-section need not be gone over at once. It can be done to the best advantage in the office. When the geologist returns from his field-work, he reconstructs from his sketches accurate sections, with horizontal measurements corrected and barometric readings adjusted.

The sketch-section illustrated on Plate II. is reconstructed in the following manner: A section-line is drawn on the map (Plate I.) along the general course of the road. Thus the first portion of this line connects O and O^4, the second O^4 and O^{10}, and in a similar manner for the rest of the section. On a very crooked road, like the one shown, some of the points, such as O^{28}, are a long distance from the line of the section; and if the dips are strong, allowance will need to be made for this fact in reconstructing the section. As a general rule, the dips are so light that a slight discordance between the road and the section-line will not affect the result; nevertheless, this point should be carefully watched and correction should be made whenever necessary.

The section is re-drawn on cross-section paper to any convenient scale, except that it is not practicable to construct it on a natural scale. Some vertical exaggeration is necessary in order to show details of stratification. The section is re-drawn on the adopted scale with the reference points, O, O^1, O^2, etc., at their proper intervals, measured on the straightened line of the section. The finished section is shown on the lower portion of Plate I.

By comparison with the note-book (Plate II.) it will be seen that the changes in structure are very slight, since with a large amount of practice the sketching can be done with considerable accuracy. The section as now constructed is still at an assumed elevation of from 2800 to 3000 feet. If the absolute elevation of any point on the section is known, the section can be corrected for sea-level; or if the elevations of several points are known, lines can be drawn on the section which will correct to a great extent the fluctuations of the barometer. Suppose the

points, O^1, O^{16}, O^{28} and O^{39}, are respectively 2030, 2280, 2150 and 1910 feet above sea-level; then we will measure down from them to some convenient datum-line, and use that instead of the assumed elevation as our standard for the section. In this case we will assume that the 1500-foot line is our datum. The elevation of the point, O^1, is 2030 feet above tide; hence it is 530 feet above the datum-line. Measure down 530 feet from O^1 and that will give a point on our standard-line. Doing the same for O^{16}, O^{28} and O^{39}, we shall have four points through which the line must pass. If the barometric readings on these points agree with the correct readings, our datum-line will be straight, but if the barometer fluctuated, the readings will vary, and the line will be broken. In the section shown on Plate I. the barometer fluctuated during the construction of the section, consequently there is considerable correction to be made between O^{16} and O^{28}.

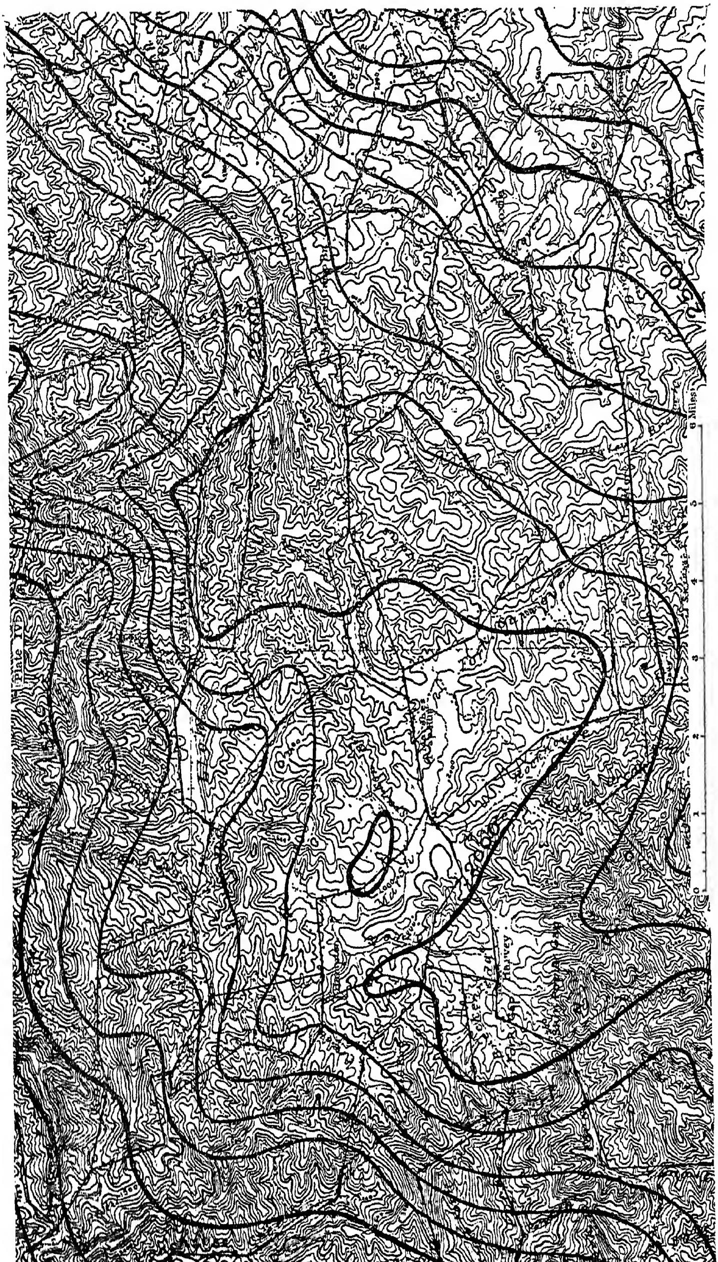
The elevation of a stratum at any point can now be determined by measuring up from the datum-line and adding that measure to 1500 feet, the absolute elevation of the datum-line. By this method all intersecting lines may be made to agree, since they are all referred to a common datum.

When this is accomplished, the geologic structure of any given area is best shown by drawing contour-lines on the surface of some prominent and persistent stratum. In the section here shown, the upper surface of any of the three beds of sandstone affords a good surface upon which to draw contours. If it is desirable to draw 100-foot contours, a scale can be run along the section holding 1500 to the datum-line, and wherever the even 100-foot divisions are crossed by the surface of the sandstone, there would be located a contour-line. By proceeding in this manner with all of the sections over a given area, a great many points will be obtained on the surface of the sandstone-bed, which, when connected by lines, will represent the attitude of the surface in question.

A contoured map prepared in this manner is shown in Plate IV. It is a portion of the Raleigh, W. Va., atlas-sheet, embracing an area of 192 square miles, mostly contained in the county of Raleigh.

On the regular topographic base-map sections, lines are ruled along each road which was traversed. This gives a connected

PLATE IV.



Map Showing Structure-Contours. Contour-interval, 100 feet.

network of sections, which, when adjusted and corrected to our common base, affords means for drawing the bedding-contours shown on Plate IV. In this particular region a massive sandstone or conglomerate, which was identified over the area shown, was selected as the most convenient stratum to contour. Each section was then gone over, and the interval between the upper surface of this sandstone and the datum-line was carefully measured. In this manner the even hundreds were marked on the section-lines on Plate IV. and then these points were connected by contour-lines. For convenience in reading the map the 1500-, 2000- and 2500-foot lines are made much heavier than the intermediate lines.

By means of such contours the geologic structure of these flat-lying rocks can be brought out more successfully than by any other means known to the writer. The most striking feature of the contours is probably their great irregularity. The 2000-foot contour varies back and forth irregularly through a zone about 5 miles in width, until near the eastern edge of the area, where it runs into a regular northwestern slope. This great irregularity of the contours is caused by slight undulations in the comparatively horizontal rocks. In mining, these slight undulations are of the utmost importance, and every operator should have a map of his property which would show them.

Mining engineers generally report that the dip averages so many feet to the mile toward the northwest or the southeast, as the case may be, but a glance at Plate IV. shows how fallacious such general statements are. True, the general dip in this case is toward the northwest; but the exceptions to the rule are so numerous that we begin to lose confidence in it and demand the actual local strike and dip of the rocks.

Such a contoured map would afford very reliable information as to the best location for mines and the direction in which the entries should be driven. If the interval between the contoured surface and any coal-seam is known, the position of the seam at any point can be easily calculated by adding this interval to the elevation of the contoured surface, when the coal-seam is above the standard bed; or by subtracting the interval when the coal lies below the datum-surface.

The section along Tug river can be reconstructed in the same

manner as the section already described; but space will not permit a detailed description here. This kind of section is especially valuable along a railroad line, where the profile of the road provides an accurately determined base-line from which to measure.

This method has been employed by me for two years in quite accurate mapping of the Coal-Measures in an area of over 3000 square miles in southern West Virginia, and by another party belonging to the same organization in central West Virginia.

Wherever tried it has been adopted, and is apparently susceptible of application wherever the rocks are approximately horizontal. In inclined rocks, its limit of usefulness is soon reached. Where dips of 5° prevail, the horizontal element of the section becomes so important that the results are not always satisfactory; and when the dips reach 10°, the system is entirely inadequate.

Note on a Shaft-Fire and its Lesson.

BY ROBERT GILMAN BROWN, BODIE, CAL.

(Colorado Meeting, September, 1896.)

There are few disasters so difficult to deal with as an underground fire. It is inaccessible at best, and generally unapproachable; and it finds most material in the very places where it can do most harm, namely, in large stopes and in "heavy" ground where timbers are massed, and in shafts and level-stations where there are inevitably strong drafts to make the burning yet more fierce.

The fire constituting the subject of this note occurred in the shaft of the Standard mine at Bodie, California, and originated after the departure of the night-force at 3 A.M., being discovered by the men going to work at 6.30. At that time it had gained already considerable headway, and was throwing smoke plentifully from the collar of the shaft.

It must be premised, for understanding of the situation, that in this shaft all hoisting is done to the 320-foot level, where the material is dropped by chutes 30 feet to the tunnel-level, through which it is run to the mill. Hoist and sheaves are all placed

underground, the upper portion of the shaft being open, but unused. The tunnel is 1800 feet long, and attempts were made at once to reach the shaft through it, but increasing smoke and gas drove the men back a hundred feet and more.

The natural course in such a case is to bulkhead everything as tightly as possible, with the hope that the fire may be smothered in its own carbonic acid, but in this instance the element of uncertainty was too large. There was a nest of old timbers in ore-chutes and shaft, above tunnel-level, where fire would lurk for months; and, more than that, there were a dozen connections through caved stopes to neighboring mines that could not be bulkheaded, as well as through openings that perhaps could. The shaft must, therefore, be approached at the ore-chutes on the tunnel-level, if possible; and this might be made possible by leaving the tunnel as the sole free intake. All other openings, to the surface direct and through other mines, were at once sealed, with the hoped-for result. In the course of two hours the chutes could be reached, and a hand-to-hand fight with the fire could be begun. In this struggle important aid was furnished, absurd as it may seem, by hand-extinguishers, and the drug-stores of the town were ransacked for bottles suitable to hold acid, while the assay-office furnished the soda for numberless charges. At first water was brought in in barrels, and a hand force-pump was used. Later, connections were made from the outside to the mine-pipe which passed out of the tunnel, and water was pumped in, to be thrown upon the fire, through a 2-inch hose, by a small centrifugal pump, driven by electricity. The latter arrangement was put into operation on the second day of the fire. By this time the chutes and shaft at tunnel-level had caved down, and the fire in the half-buried timbers had to be fought for a day or two more, the fear being that it would work down the shaft.

Meanwhile water was being pumped from a neighboring mine through fire-hose into the collar of the shaft, which spattering and spraying down finally saved the upper 200 feet of it.

The damage was ultimately found to extend upwards from tunnel-level, where the cage had wedged, and to include the chutes and hoist-station. This must be regarded, in view of the much greater disaster threatened, as a most fortunate and satisfactory result.

By way of deducing for general application the lessons taught by this experience, it may be well to classify the elements which combined to lessen the injury incurred under the two heads of existing conditions and active relief-measures.

Existing Conditions.—These might well be called adventitious or accidental circumstances, since they can scarcely be said to have been provided deliberately with a view to the emergency in which they came into play. Apart from the conditions involved in the nature of the mine, the locality of the fire, etc., they comprise:

a. The wedging of the cage just above tunnel-level.

b. The presence of a water-pipe in the tunnel, with easy outside connections to adequate water-supply.

c. The possession of a centrifugal pump and electric motor that could be put into operation almost immediately.

d. The supply of water from a neighboring mine, and sufficient hose to bring it to the collar of the shaft.

Relief Measures.—Under this head may be mentioned:

e. The bulkheading of all approaches save one.

f. Fighting the fire with hand-extinguishers.

g. Hauling in water and using hand force-pumps.

These two categories are not food for pleasant consideration, yet it is safe to say that the conditions of 80 per cent. of the metal-mines of the West are to-day no better.

It is clear that the first list, which really made the fight a winning one, deserves special consideration, with the view of converting its elements into relief-measures for future application.

a. Some years ago a fire-door, with automatic release, was devised for use in shafts, and actually employed in several instances.* It was 8 inches thick and iron-sheathed, and was designed to save a shaft from burning with the shaft-house. Some such device, placed between levels as well, would be invaluable in isolating various portions of the shaft.

b. Where adit-tunnels exist, outside connection to water-supply should be provided and fitted with couplings and hose, if the conditions are such that in a possible contingency this supply may be required.

c. If pressure is not ample for this purpose a small hand

* See *Eng. and Min. Jour.*, April 7, 1894.

force-pump (about 3-inch barrel), or, if there is electric power in the mine, a 3 K. W. motor with 12-inch centrifugal pump should be at hand.

d. For surface water-supply a main should be laid, when practicable, from some neighboring mine, with such connections to pumps and hydrants that water can be furnished by either mine for the other's need. This seems to be a fair case for co-operation among several adjacent properties, lessening the expense and increasing the efficiency of protection against fire.

In the case here under consideration relief-measures would have been more effective if the means had been more promptly at hand, and hence the following observations:

e. Underground connections with other mines are to be avoided; but where they exist they should be fitted with heavy air-tight bulkheads, with passage-traps if necessary. Main air-ways should be supplied with tight doors. In stoping close to boundary-lines it should be a first principle that a solid pillar of rock between mines is the best of protections, since ordinary filled ground will leak enough air to keep a fire alive for months.

f. A hand-extinguisher should be kept at every dry station.

g. A tank-car and a hand-pump should be at hand for use in tunnels or levels.

A few hundred dollars invested in simple provisions of this kind would serve as a most efficient insurance-premium.

Mr. Channing has written adequately* of the advantages of close bulkheading, which will unquestionably remain the standard method of fighting underground fires; but there are cases, like the one here described, in which the complete exclusion of air is not practicable. In such cases the policy outlined above is worthy of consideration, namely, that of choosing the best point of attack and partially bulkheading, so that fire and gas may draw away therefrom, making approach possible from that quarter. At the time of this writing, exactly a month after the breaking out of the fire, the shaft, from 15 feet above tunnel-level downwards, is open and is operated by a temporary hoist; the old hoist-station, which had caved full, has been opened; the machinery has been removed and is now undergoing repairs, and for three weeks ore-mining has been recommenced in the upper levels, the product being handled through improvised

* *Eng. and Min. Jour.* Jan. 16, 1892.

chutes. With attempted bulkheading we should have been entirely excluded from the mine, and should be to-day but just beginning to be able to estimate our loss.

Additions to the Power-Plant of the Standard Consolidated Mining Company.

BY ROBERT GILMAN BROWN, BODIE, CALIFORNIA.

(Colorado Meeting, September, 1896.)

The original power-plant of the Standard Company has been described* by Mr. T. H. Leggett, late president and manager of the company; but since the presentation of his paper considerable additions have been made, which will be found interesting because of the engineering features involved and of the attainment of a high degree of flexibility with the inflexible, two-wire, alternating system of electrical transmission.

Although these additions have been completed during the last year, they constitute, to a large extent, only the carrying out of plans that belonged to the original conception of the installation.

The regular operation of the water-power through the winter proved very difficult; there was a considerable length of ditch to be kept open, and, as the stream is small and any freezing night may stop the whole flow of water, there was the additional trouble of having no reserve to draw upon. The rule for the winter is, that after a night of hard frost water begins to move in the cañon before noon on all but the coldest days.

To obviate the nuisance of the anchor-ice, and to make available all of the water, a dam of sufficient height to impound several days' run of water has been built across the cañon, 3875 feet above the power-house, and provided with pressure-pipe well below the surface. In place of the 1571 feet of pipe-line from penstock to power-house, there has been substituted a line 3875 feet long to bring water direct from the dam.

The dam is of logs, cribbed up in squares of about 12 feet, ballasted with earth and rock and sheathed on the water-face with 3-inch plank.

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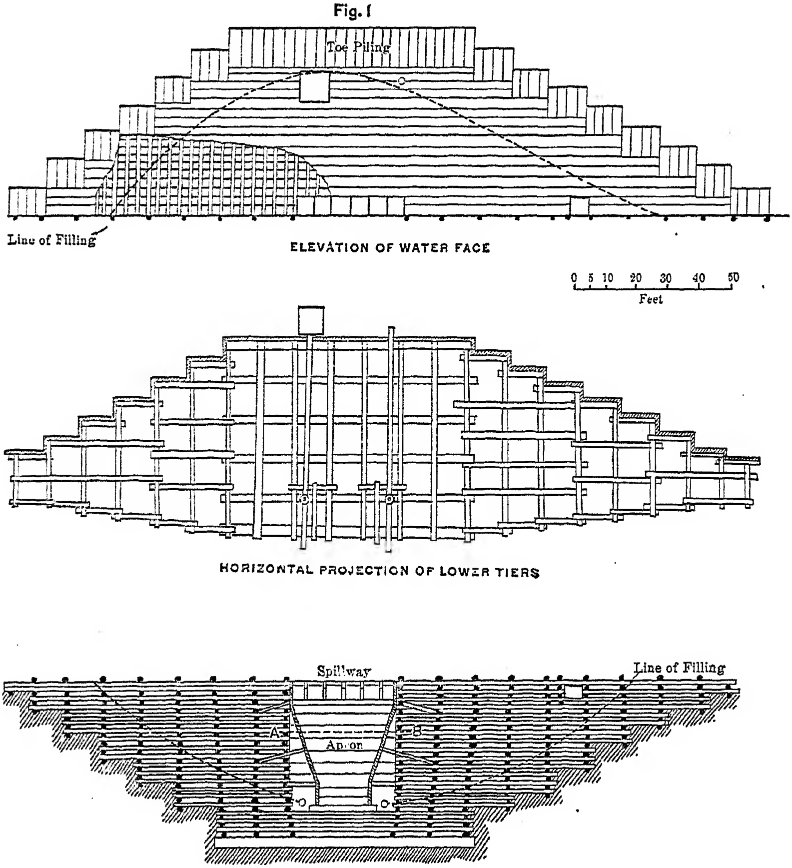
* *Trans.*, xxiv., 315.

Ample precautions have been taken against the working of water around the ends and bottom by the use of toe-piling, sunk to, and driven into, firm cement-gravel, and tamped in place. The toe-piling is 10 feet long and buried under 10 feet of rock-and-dirt-filling, piled in along the ends of the dam.

Fig. 1



ELEVATION OF FREE FACE

Green Creek Dam.

What may be properly called the king-log of the whole construction is the large one shown in Fig. 1, at the bottom of "Elevation of Free Face." It is 80 feet long by more than 3 feet in diameter, and, to a certain extent, it bears the whole thrust of the water. From this, at the lowest, truncated apex,

the dam lengthens out by horizontal steps and terraces, excavated in the solid gravel, to an extreme top-length of 230 feet.

These and all of the other principal details of construction can be studied in Figs. 1 to 3; but the process of construction demands a few words.

After the first few courses had been placed, excavation for the benches went on side by side with building, the material removed by the one process being employed for the other. As greater height was attained and more ballast was required, double car-tracks were laid along the dam, skirting the bank, up and down stream where a breast could be attacked by undermining and caving.

The order of procedure was as follows: On the completion of a single bench-level of seven or eight tiers the track was removed and the next bench-level was carried up on the west bank (right-hand bank in Fig. 1) and out to the length of single logs—perhaps 30 feet. On this the track was again started and filling was resumed, while the building-gang advanced independently.

The rafters for the sheathing were carried up on the water-face as fast as the work allowed, and were closely followed by the sheathing itself. This was carefully caulked with tamarack bark, which is an excellent substitute for oakum, and can be relied on not to work out.

Particular care was taken with the ballast placed against the sheathing; clayey soil was used, free from rocks, and tamped against the plank.

In joining the logs only sufficient hewing was done to insure firm contact. All cross-joints were secured by iron drift-bolts, ranging from $\frac{5}{8}$-inch to 1 inch in diameter. In exception to this the few last tiers were pinned with 2-inch split-tamarack pins.

As the filling was from the west bank the logs were taken from the east, being handled by a winch and snatch-block. Had the conditions permitted, it would have been easier to keep the water near the top of the work and float the logs down against it, only using the winch for their final placing.

The spill-way, 35 feet wide, with an extreme depth of 5 feet, cuts the dam over the center of the stream (Fig. 1). It is lined on bottom and sides with plank, and the floor stands 28 feet

vertically above the top of the strainer. This height can be increased by sectional gates to 32 feet, or 1 foot below the top of the dam.

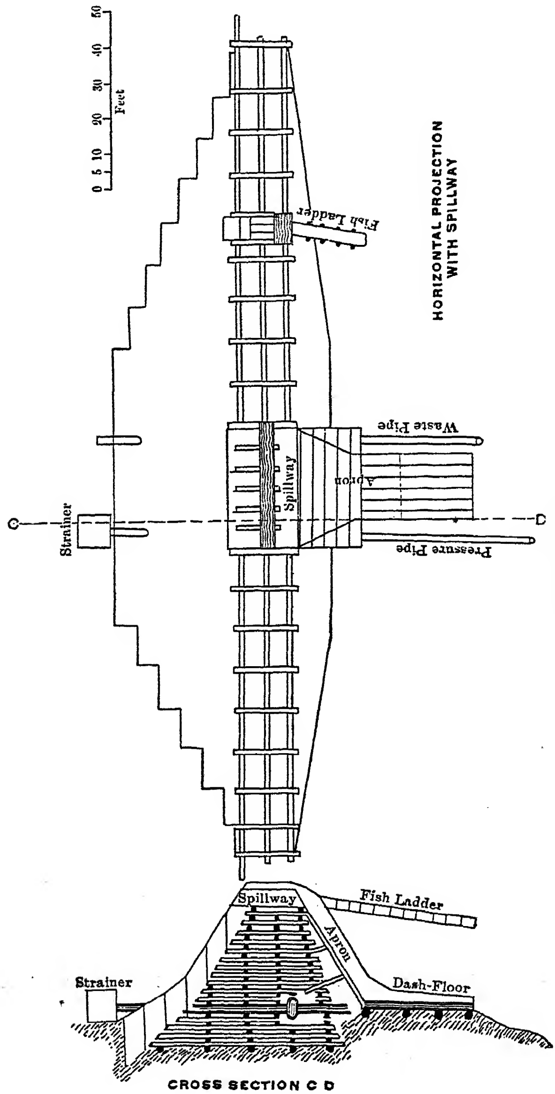


Fig. 2

Green Creek Dam.

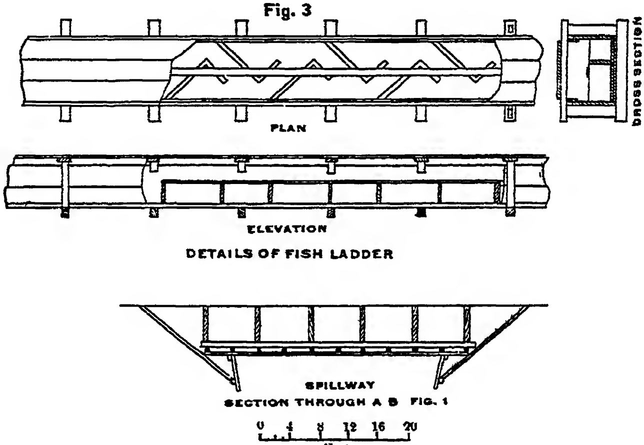
The apron, onto which this discharges, contracts from the full width at the top to 18 feet at the bottom, with a slight inward

batter of the wings, so as better to hold the splash (Figs. 1 and 3).

Below this a dash-floor of logs catches the force of the overflow (Fig. 2). This is one of the crucial points of the construction, and the greatest care was taken in laying it, so as to prevent under-washing of the dam. Selected straight logs were laid with hewn joints on four transverse mud-sills and secured with drift-bolts.

The pipe-line pierces the dam on one side of this floor and on the same level; and on the other side, about 2 feet lower, is the waste-pipe (Fig. 2). The former is protected at its upper end by a coarse wooden strainer, 7 feet cube, inside of which

Fig. 3



Green Creek Dam, Details.

is ½-inch-mesh wire screening, the latter being intended principally to keep out twigs and tamarack-cones. Anything that passes this screen will pass the nozzles at the wheels.

The fish-ladder (a requirement of the State law) is shown in detail in Fig. 3.

Within the body of the dam are two chambers for the 24-inch gate-valves in the power and waste-pipes. These are thoroughly protected from frost by double doors, opening under the apron.

The precautions taken against leakage by toe-piling and caulking have proved very effective. Scarcely a trace of seep-

age can be found below the dam; and the valve-chambers, 30 feet below the surface of the water, are dry.

In the following tables are collected the most important data relative to the construction:

Green Creek Dam.

Dimensions of Dam Proper.

	Feet.
Length, bottom,	80
" top,	235
Thickness, bottom,	60
" top,	15
Height,	42
Batter, water-face, lower portion,	1 in 1
" " " upper "	1 " 2
" free face,	1 " 4
Extreme height,	50

Dimensions of Spill-Way.

	Feet.
Height of floor,	37
Length (across stream),	35
Width (through dam),	20
Depth,	5

Dimensions of Apron.

	Feet.
Width at top,	35
" " bottom,	18
Length down slope,	33
Depth (height of sides),	5
Slope, from vertical,	1 in 2

Dimensions of Dash-Floor.

	Feet.
Length,	30
Width,	18
Depth,	5

Dimensions of Fish-Ladder.

	Feet.
Length,	289
Width,	3
Depth,	2
Slope from horizontal,	1 in 7
Number of steps,	109

Summary of Material.

Dam Proper.

Sawn lumber,	49,000 feet, B. M.
Toe-piling,	472 pieces
Logs,	908 "

Spill-Way and Apron.

Sawn lumber, ... 4000 feet, B. M.

Fish-Ladder.

Sawn lumber, ... 12,000 feet, B. M.

Cost of general supplies for whole work, about $1000.

Ratio of labor to all other expenses, ...	2:1
Average number of men on dam, ...	33
Labor, total number of shifts, ...	6000
Average number of men on pipe-line, ...	11
Labor, total number of shifts, ...	2000
Total time of construction, ...	26 weeks.
Stoppage of plant for changes in pipe-line, . . .	18 days.

The stability of the dam is demonstrated in Fig. 4, the broken lines representing an addition of 10 feet to the height of the structure. The solid lines show the pressure-lines of the present dam. The principal data given on the stress-sheet all indicate insignificant pressures as compared with the resistance; but it must be borne in mind that the regular stresses are the least that are to be borne, the extraordinary ones of flood and mountain cloud-burst being always among the possibilities. It was with the view of guarding against these that the more generous proportions were adopted.

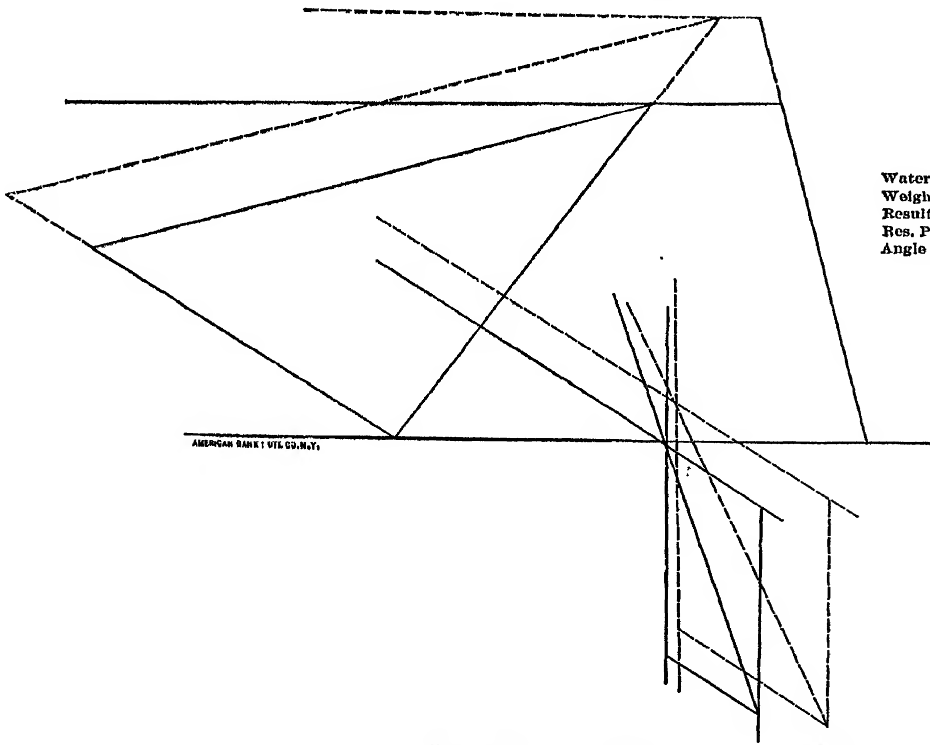
The initial length of the pipe-line is a 20-foot taper piece, 30 to 24 inches in diameter, the latter being the size of the head gate-valve.

The diameter further decreases with increase of pressure. The pipe is all double-riveted, dipped and provided with slip-joints to within 650 feet of the station, where collar-and-sleeve lead-joints are used. It is laid in a trench, much of which was rock-cutting, and is covered with 12 to 18 inches of earth. Along its length there are seven air-valves to prevent collapse.

Table of Pipe.

Diameter. Inches.	Thickness of iron.	Length. Feet.
30 to 24,	No. 14,	20
24,	" 14,	1193
22,	" 16,	365
22,	" 14,	1193
20,	" 14,	482
18,	" 14,	655
Total,		3908

Fig. 4



DATA

	40 Ft.	50 Ft.
Water Pressure	306 T.	493 T.
Weight of Dam Slice	616 T.	663 T.
Resultant Pressure	825 T.	1025 T.
Res. Press. per sq. Foot	2700 lbs.	3361 lbs.
Angle Res. Press. with Horizon	78°	66°

Scale of Dam, 1 Inch = 20 Ft.
Scale of Stresses, 1 Inch = 500 Tons

Stress Sheet of Ten-Foot Sluice, Middle of Dam.

The vertical height obtained is 342.8 feet, representing a static gauge-pressure of 150 pounds, and the kinetic head, calculated from the observed running pressure of 146 pounds, is 334 feet, giving a loss from pipe-friction of 8.8 feet. It may be noted here that in this there is an excess of 3.7 feet above the theoretical loss of head due to friction in the pipe. It is quite possible that this results from the position of the gauge on the pressure-drum, which is such that the eddies, due to the enlargement, would there be greatest.

To increase the power delivered by the generator, the speed has been increased from 870 to 990 revolutions, and to give the wheels greater resilience against sudden load, the nozzles have been enlarged to $1\frac{1}{2}$ and $1\frac{5}{16}$ inches. There are some drawbacks to this change of nozzles, as the efficiency of the water is seriously impaired by running under ordinary load with the butterfly regulating-valves partly closed; and in time of low water, it has been necessary to revert to the smaller nozzles. But even in this case the drop of speed, at a sudden increase of load, is only to be told by the drop in the hum of the machine and a slight wave in the incandescent-light circuits. Possibly the retard would amount to 4 per cent. for ten seconds.

As an offset to this, at the Bodie end of the line, the $5\frac{1}{2}$-ton fly-wheel of the mill-engine has been belted on to the line-shaft, lessening materially the drop due to the sudden throwing on of the mine-hoist.

The question of a suitable governing-valve for high-water velocities is an absorbing one. The ideal valve would be one that diminished the volume passing (*i.e.*, the cross-section of the pipe) without increasing the loss of head due to friction.

Such a valve is probably not mechanically attainable, and of the three common kinds, the gate-valve, the plug-cock and the throttle (butterfly), the last seems to be the best for large, the gate for medium and the plug-cock for very small ranges. This will be seen from the following table, prepared from Weisbach's tables on valve-resistance,* by proportional interpolations. The degrees given in this table belong to the plug-cock, and represent the angle with the axis of the pipe; the area-ratio is that of the valve-opening to the area of the pipe.

* See Coxe's translation of Weisbach's *Mechanics of Engineering*, 7th ed., pp. 900–902.

Coefficients of Valve-Resistance.

Angle	5°	15°	25°	35°	45°
Area-ratio	0.926	0.772	0.613	0.458	0.315
Plug-cock	0.05	0.75	3.10	9.68	17.3
Gate-valve	0.12	0.67	2.02	6.19	17.
Throttle.	0.20	0.75	2.08	5.09	16.

Suggestive as this table is, too much stress must not be laid upon it; such results clearly depend largely upon the proportions of the individual valves used in the determination.

It can be added that, assuming the above as fairly correct, the loss in each nozzle, at the ordinary position of the butterfly, is more than 6 feet in head, or a total of 25 feet for the four wheels.

The Doolittle differential governor is very delicate in its adjustment of the speed, and it is interesting to watch the parallelism between its motions and the rise and fall of the ammeter. As a regulating device, it gives complete satisfaction.

As first constructed, the receiver was anchored against the thrust of the water by wire cables; but such supports proved not rigid enough, allowing, by their contraction and expansion, sufficient motion to throw the nozzles out of line with the wheels. To obviate this defect, the receiver has been cased in a timber frame or crate, with longitudinal and transverse bolts, and the whole braced, by long struts at top and bottom, against two deeply-buried "dead-men," which effectually take the thrust.

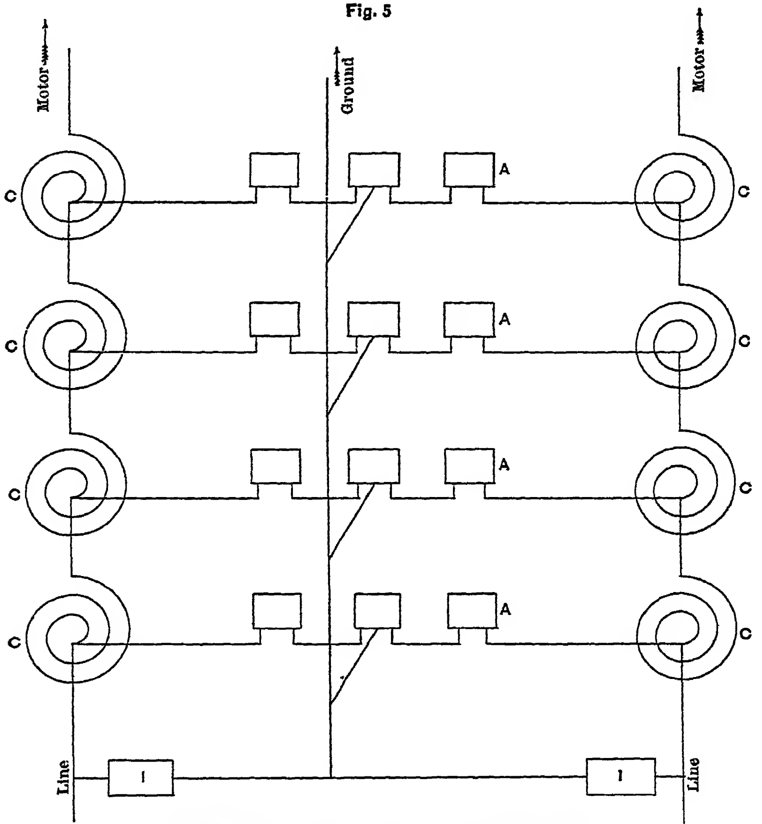
The only change in the electrical part of the generating-station has been replacing the fuller-board cells for the armature-coils by pure mica-cells, which heavy form of insulation has left the Bodie end of the line the weaker, and it is at that end now, if at all, that the troubles from lightning occur.

In this connection a few words must be given to the annoyance suffered from lightning, and the precautions taken against it. Mr. Leggett has discussed the subject fully in a paper published in the 12th report of the California State Mineralogist,

1894, but in his paper before the Institute only passing reference was made to it.

The safeguards adopted at Bodie are substantially those outlined by Mr. A. J. Wurts,* as installed with satisfaction by himself at Telluride, Colo.

Fig. 5



A, Arresters; C, Choke Coils; I, Inductive Resistances.

Arrangement of Lightning-Arresters.

Twelve 1000-volt non-arcing lightning-arresters, with 18-inch choke-coils, have been placed in an arrester-house, outside of the station, at each end of the line, and connected as shown in Fig. 5. The "ground" consists of a copper-plate of 6 square feet area, with fine charcoal above and below. At the receiv-

* *Trans. Am. Inst. of Electrical Engrs.*, vol. xi., p. 337, May, 1894.

ing-end, where the ground is dry, moisture is supplied by a small water-pipe. In addition to these, there are three 1000-volt station-arresters on the switch-board, and the line-wires have at their terminals inductive resistances (Fig. 5) for the gradual discharge to ground of static accumulations on the line, if there be any such.

The theory under which such a protection is provided is that the lightning-discharges, being highly oscillatory, are dammed back in the choke-coils by their own inductive resistances and leap the gaps in the arresters in place of passing on to the machines, while the power-current, of comparatively few oscillations, is but faintly affected. By connecting these coils in series the end is served of repeatedly "sifting the siftings," a phrase that becomes clear if each coil be viewed as a rather imperfect screen for the sifting out of thunder-bolts. It is possible also that having the protection at several adjacent points of the line is an advantage, as there seem to be shifting nodal points, from which the storm-discharge cannot be drawn. The existence of these is only surmised, and their law is totally unformulated; but it is obvious that several points of possible discharge lessen the probability of the arrester-mechanism being connected at a nodal point.

To what degree these arrangements protect the machinery can be best understood after a visit to the arrester-house during an electric storm, when the discharges will be grounding through the arresters at the rate of one a second for a period of from 5 minutes to ½ hour. But not all of them go that road, and sometimes a bolt will puncture the insulation on several coils at once. Thus the first storm of the summer season of 1896 broke without warning, and, almost at the first flash, short-circuited five armature-coils in the motor, and a day or two later, out of a moderately cloudy sky, a bolt came in, passing all the arresters and punctured another coil, and this while the arresters were freely snapping.

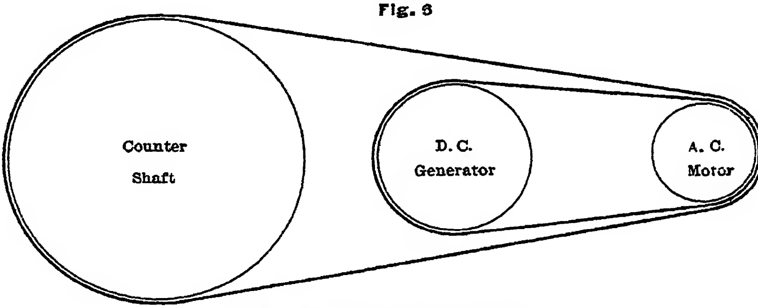
The term "bolt" has been used, but as a matter of fact it is not generally believed that the "lightning" which troubles the machines is more than the inductive result of heavy discharges between cloud and cloud, or cloud and earth, or perhaps results even from the proximity of electrified vapor masses to the line. Probably a genuine bolt would make itself felt in a more

serious way than the puncturing of a few millimeters of insulation.

The only addition to the alternating-current plant has been the "exciter," placed in the motor-room, so as to do away with the sliding contacts on the commutator, which were a continual source of dissatisfaction. This is a four-pole, D. C. 65-volt, 1300-r.p.m. generator, belted by a specially-constructed pulley to the motor-shaft.

The prime cause for increasing the power, as described above, was the extension of the plant to include hoisting and pumping operations in the mine. In the summer of 1894 the hoisting-works of the company were completely destroyed by

Fig. 6



Arrangement of Driving-Belts.

fire, and it was then decided not to rebuild, but to put in an underground hoist and send out all rock and ore through the tunnel, about 350 feet below the collar of the shaft.

To accomplish this a 90-K. W., 500-volt, direct-current, multipolar generator was installed in the motor-room at Bodie, driven with an under-belt from the main A. C. motor-pulley, as shown in the diagram, Fig. 6, and also very clearly in Plate B. This method of distributing power from one pulley has proved very effective, and was in this case the only way of directly operating the additional machinery. Plate B gives also a partial view of the direct-current switch-board. The automatic circuit-breaker and double jaw-switch are the most prominent features on this board; but between the lower belts can be seen the wheel of the regulating resistance. It is further fitted with volt- and ammeter and lightning-arresters. The circuit-breaker is of the General Electric Co.'s make, with

a carrying-capacity of from 300 to 500 amperes, adjusted to open the line automatically in case of a dangerous flow of current. It has successfully come into operation on more than one short-circuit in the line and protected the machines.

The current from the D. C. generator is carried by four No. 1 bare wires 1200 feet to the tunnel-mouth, and thence by two No. 000 covered cables 800 feet to the shaft. Here there is a further connection by No. 0 covered cables for 500 feet, down to the pump. From the tunnel-mouth there is a 600-foot branch of two No. 1 bare wires to the tailing-plant, where a small amount is used for lighting (five 100-volt lamps, connected in series) and to run a 3-K.W. bi-polar motor.

There are no data for an exact analysis of the work of the various D. C. machines; but from the hourly record of the motor-men the following has been averaged:

	Horse-power.
Lighting and line-loss,	3
Tailings-plant motor,	3
Mine-hoist,	33
Mine-pump,	31

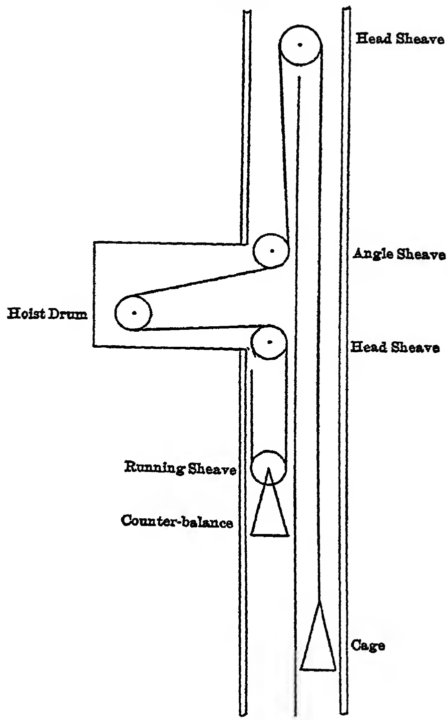
The interest of these figures lies chiefly in the distribution of the work indicated. It is to be noted also that the pump and hoist are never run together, as pumping is only resorted to at intervals, when water is needed at the surface-plants.

First in importance of the D. C. motors is the mine-hoist. This is placed on tunnel-level, 350 feet below the collar of the shaft, and operates from below the 800-foot level to the 240-foot level, the landing-station for ore and waste being 30 feet above the tunnel.

The hoist is an ordinary small Lidgerwood geared, and fitted with the double conical friction-clutch of that make. On the same bed is mounted a General Electric Co. 50 H.P., P.W. iron-clad motor, acting through two reduction-gears, with a proportion between extremes of 14 revolutions to 1. This gives to the drum a maximum of 30 revolutions and to the rope a linear velocity of 300 feet per minute. But with loaded cage this becomes 240 feet per minute. The plain iron drum has been divided into halves by a wooden flange, bolted through the shell and stiffened by an iron rim, to permit the use of a counterbalance-cage. The rheostat is of type 83, General Electric Co.'s make.

Fig. 7 shows in diagram the arrangement of sheaves, station, shaft, etc. Particular attention should be paid to the counterbalancing device. It is obvious that with this arrangement of sheaves, which is the one demanded by the situation, the counter-balance cannot have equal travel with the cage. The running sheave eliminates this difficulty, and the only elements

Fig. 7.



Counterbalanced Hoist.

of the arrangement that suffer are the rope, which turns over a rather small sheave, and the counterbalance-cage, which carries double weight to compensate for half-travel. The latter is, of course, easily allowed for by the construction of the cage, and the former is of comparative insignificance where, even under such unfavorable conditions, the cost of rope amounts to less than one cent per ton of ore handled.

The counterbalance is adjusted so as to nearly equalize the

work of the motor on up- and down-trips, with the consequence that it is heavy enough to pull up the empty cage. The advantages of such a balancing are to be seen with particular clearness in the present instance, where 30 H.-P. for A hours can be better spared than 50 for $\frac{A}{2}$ hours; that is, where the limit of the available power is nearly reached.

The fire of April, 1896, which has been made the subject of a few notes for the Institute (*ante*, p. 315), attacked this part of the mine, and no better proof could be adduced of the satisfactory operation of the arrangement above described than the fact the whole is now being restored to practically the original shape.

The pump is a Dow triple-plunger working under the considerable head of 520 feet, and driven by a quadri-polar 30 H.-P. motor, working 600 revolutions per minute and reducing by two sets of gears to 52 strokes for the plungers. The motor-pinion is of raw-hide, which was chosen for its insulating qualities, but has proved itself unsuitable for occasional use, as in this case, in a damp atmosphere. A couple of days of idleness causes the layers of hide to bulge out irregularly, increasing largely the friction, and an enforced disuse of several weeks, as at the time of the fire, has made it useless.

At the customary speed of 46 revolutions (= 138 plunger-strokes) per minute, the pump delivers 108 gallons per minute, with an expenditure of 29 H.-P. It is interesting to note in this connection that the actual work of raising the water, exclusive of friction, is 17 H.-P., leaving what seems the extreme amount of 12 H.-P. as absorbed by machinery and water-friction.

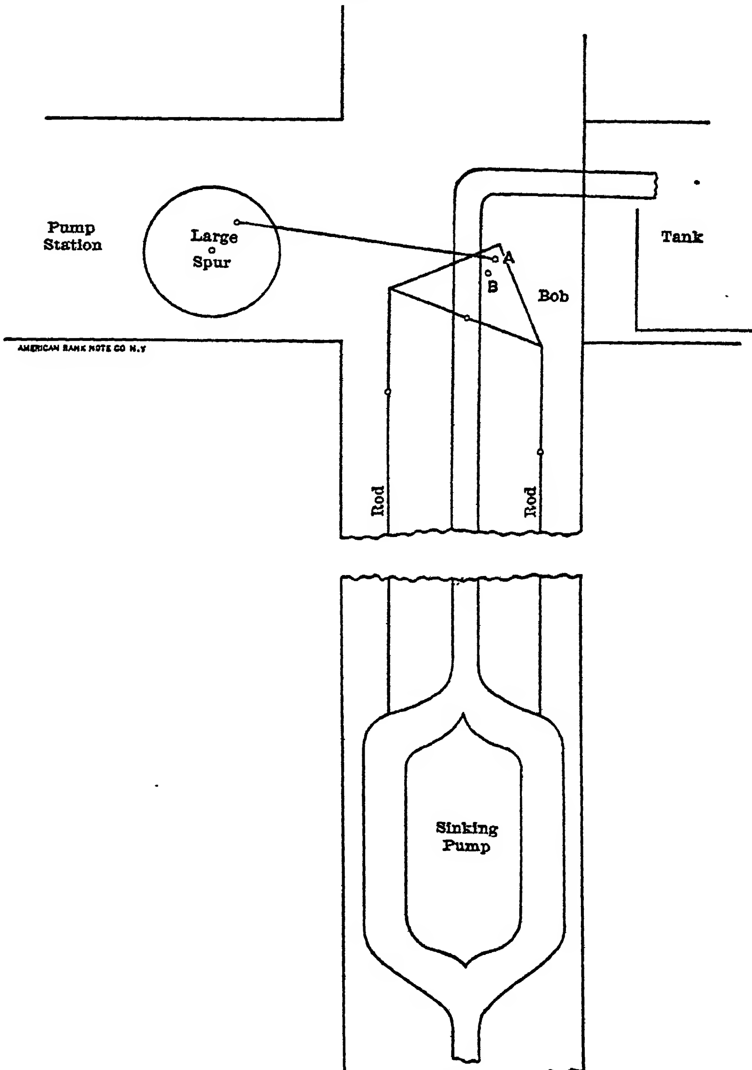
There is a sinking-pump attachment to the station-pump, which acts through the medium of the large, small-toothed gear, to be seen fronting the man in plate C, and in Fig. 8 as well. The "sinker" is hung at the surface of the water, 40 feet below the station-floor. It is a double-bucket lift-pump, connected, as shown in the diagram, to the acute angles of the small iron bob set in the shaft, which in turn acquires its motion from the large spur through a connecting-rod.

As a means of adjusting the water raised by the sinking to the needs of the station-pump, a limited modification of stroke is provided for by the two pins, A and B, at the obtuse angle of the bob, to either of which the pitman can be attached.

The rods are 1¼-inch round iron, running in wooden guides,

and the water discharges into a tank, from which the main pump draws, but does not lift.

Fig. 8



Sinking-Pump.

The whole sinking-pump device is admirable in design and operation.

In closing, it will not be amiss to touch briefly upon the salient points of the administration of the plant.

All the machinery is under the care of a competent electrical engineer. At the power-house there are two men and a cook; at the motor-room, two men, and at the hoist two men—a total of six men and one boy (cook) for 24 hours' work; in addition to which a pump-man is taken from other work in the mine for about half a shift daily.

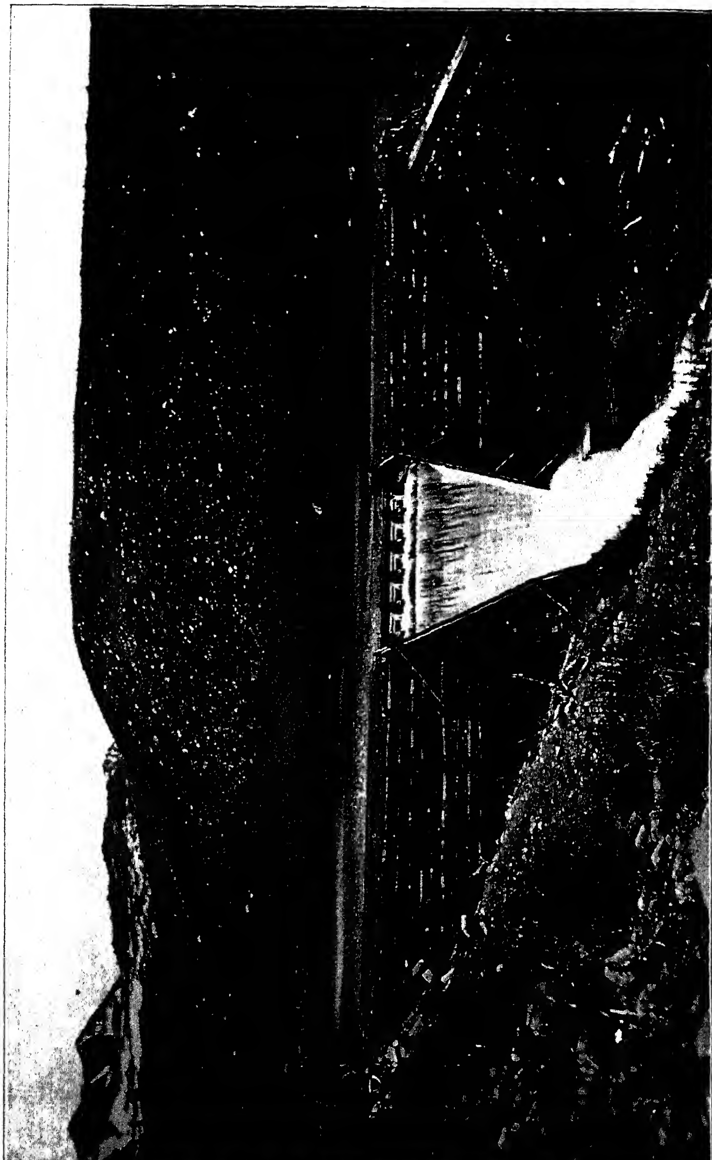
About once in three weeks the whole plant is stopped for a couple of hours to change oil in the boxes, examine belts, look over the machines, test for grounds, etc.; but several continuous runs of over four weeks have been made.

An hourly record is kept at both power-house and motor-room of the readings of the various instruments, which serves to keep the men up to their duty as well as to furnish tolerably accurate data of the work of the machines. At the pump a similar record is kept of hours run, strokes of pump and gallons of water raised, and in the office, as a final check on work at all parts of the plant, there is a recording Bristol voltmeter on the high-tension circuit.

Considerable space has been given in the foregoing pages to difficulties and defects, under the conviction that their study is their antidote.

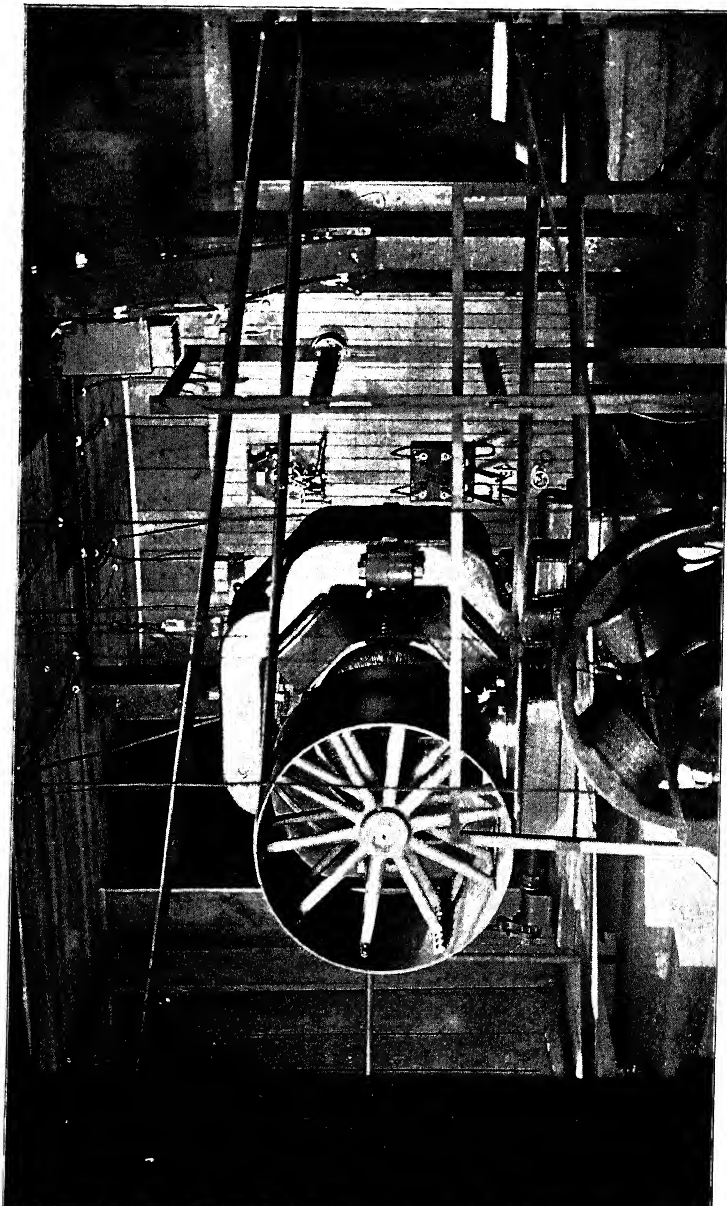
In conclusion, the writer would say that a certain familiarity with the original plant has of necessity been taken for granted, and that any obscurities can be made clear by a study of the remarkably interesting and valuable paper of Mr. Leggett, to which reference has been more than once made. To that paper the present is hardly more than an appendix, just as in great degree the portions of the power-plant which it describes were but the completion of the originally conceived plan. The electrical additions were all installed under the advice and supervision of Mr. Fred. H. Davis, Electrical Engineer of the Westinghouse Co.

PLATE A.



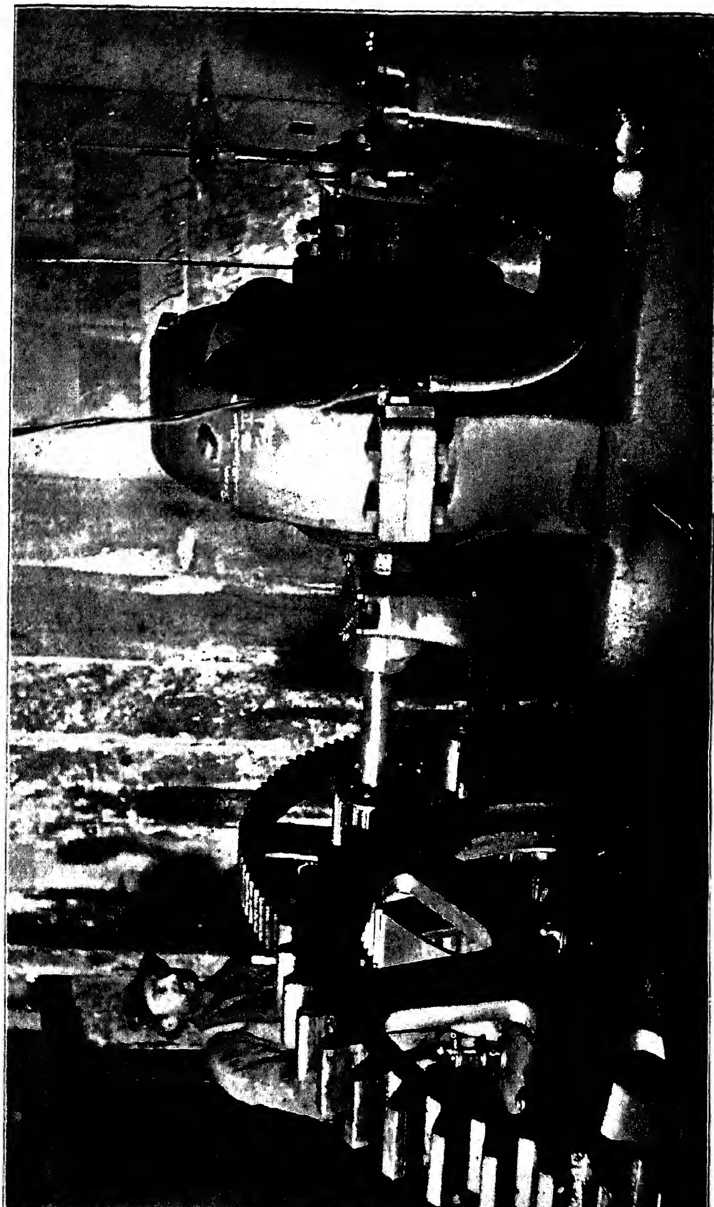
Green Creek Dam.

PLATE B.



Direct-Current Generator.

Plate C.



Mine-Pump Motor.

The Newton-Chambers System of Saving the By-Products of Coke-Manufacture in Bee-Hive Ovens.

BY ROBERT A. COOK, NEW BRUNSWICK, N. J.

(Pittsburgh Meeting, February, 1896.)

In the increasing struggle to cheapen the cost of our iron and steel products a great deal of attention has been given to economies in the manufacture of coke; and while but little has been accomplished in this direction in the United States, the narrow margin between cost and market-price, which has obtained here for the past few years of continuing manufacturing depression, has brought the coke-maker face to face with the problem, how to save, and benefit by it, the hitherto waste products of distillation.

These may be summed up under three heads, generally ranking, in value, in the following order:

1. Ammonia salts.
2. Oil- or tar-products.
3. Gas.

There are cases in which the last may assume the position of greatest importance.

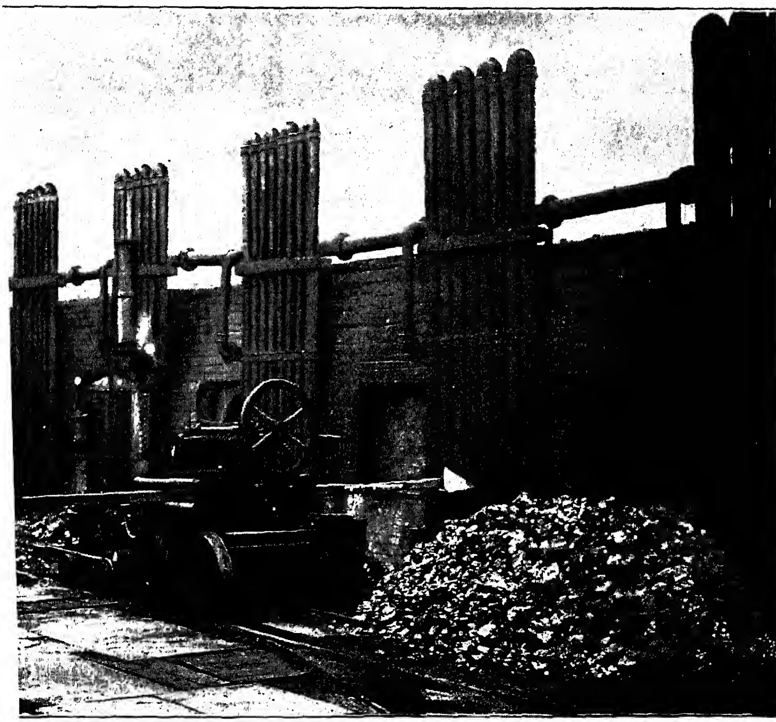
The greatest obstacle encountered by the American coke-maker, on investigating the subject, has been the apparent necessity of discarding his existing plant of bee-hive ovens, and erecting an entirely new and very expensive system of retort-ovens, against the product of which he has probably been prejudiced from his earliest knowledge of the subject.

The object of this paper is to bring prominently before our coke-experts a brief history of the development at Sheffield, England, of a most successful system of the utilization of by-products in connection with an already existing plant of bee-hive ovens. This system has now been in satisfactory operation for about five years, and is applied to a series of blocks of ovens, comprising about 100 in all.

At this plant, belonging to the well-known firm of Newton,

Chambers & Co., Limited, Thorncliffe Collieries, Sheffield, the gas from the ovens is utilized for generating all the steam used about the works, and the oil and ammonia are recovered. One great advantage is that if any hitch should occur in the by-product-plant, the ovens can be operated in the usual way, and not the slightest delay or irregularity need occur in the manu-

Fig. 1.



Thorncliffe Coke-Ovens.

facture affecting the quantity or quality of the coke produced. In fact, the saving of the waste product is purely an adjunct to the operations of the coke-plant.

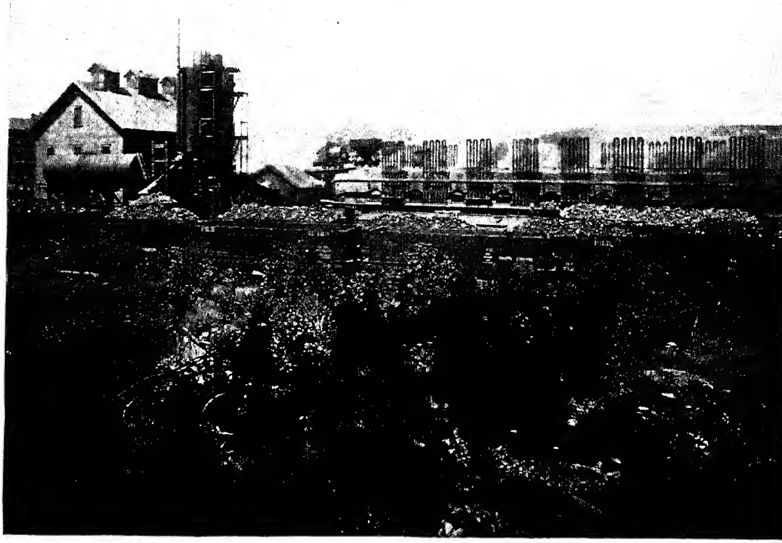
The Thorncliffe plant, shown in Fig. 1, consists of ordinary Yorkshire bee-hive ovens, 11 feet in diameter, by $8\frac{1}{2}$ feet high. They are more carefully built than is usually the practice in this country, the fire-brick being backed in a 4-inch course of red brick, and the spaces between the ovens being concreted

instead of rammed with clay. The ovens are thus more likely to be gas- and air-tight, and have probably twice as long a life.

The following are the attachments constituting the system:

1. A fire-clay pipe is built into the crown of the oven, passing completely around the dome, close to the inner lining and near the spring of the arch. One end passes to the outside of the

Fig. 2.



Latrobe Coke-Ovens.

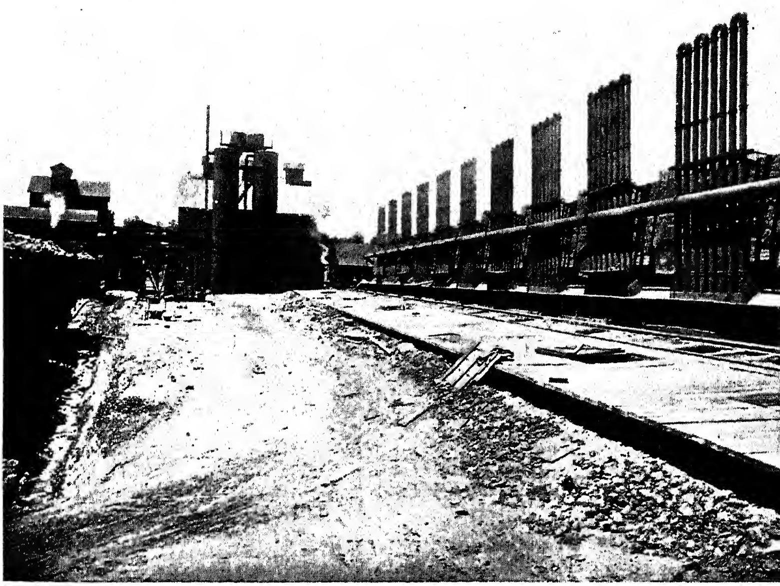
oven, where it connects with an iron blast-pipe, or main; the other end, entering the oven above the charge, furnishes a supply of heated air required for the coking operation.

2. An opening, left in the bottom or floor of the oven, near the door, communicating with a cast-iron hydraulic main, for cooling purposes, serves to take off the liquid and gaseous products of distillation. To the hydraulic main is attached an oil-drip, which separates the heavier liquids; and the gas passes

on through the surface-condensers, consisting of light cast-iron pipes, set close to each oven. In these condensers the further separation of oil and liquor from the gas takes place, and everything passes thence to a 24-inch main leading to the settling-tank.

3. The gas passes on from the upper portion of this tank to

FIG. 3.



Latrobe Coke-ovens.

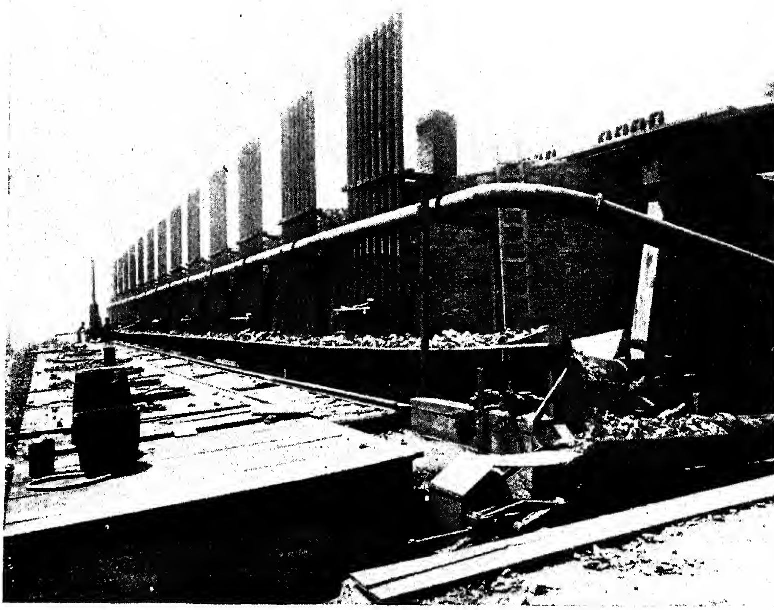
the boiler-house, where it heats 8 large Lancashire boilers, and is conveyed thence to a stack 150 feet high.

Before the introduction of this gas for fuel, the weekly consumption of coal under these boilers was 200 gross tons. This has been entirely done away with; and, as the gas is the waste product of ovens using 1000 tons of coal per week, the saving is equivalent to 20 per cent. of the coal coked. These boilers have been running steadily without the intervention of a gas-

holder or other pressure-regulator since the attachment was first made.

4. An elaborate system of tanks and stills has been developed at Thorncliffe for the complete separation of the various oil-products. After taking off the last traces of ammonia liquor, these oils are found to give several grades of high-solvent

Fig. 4.

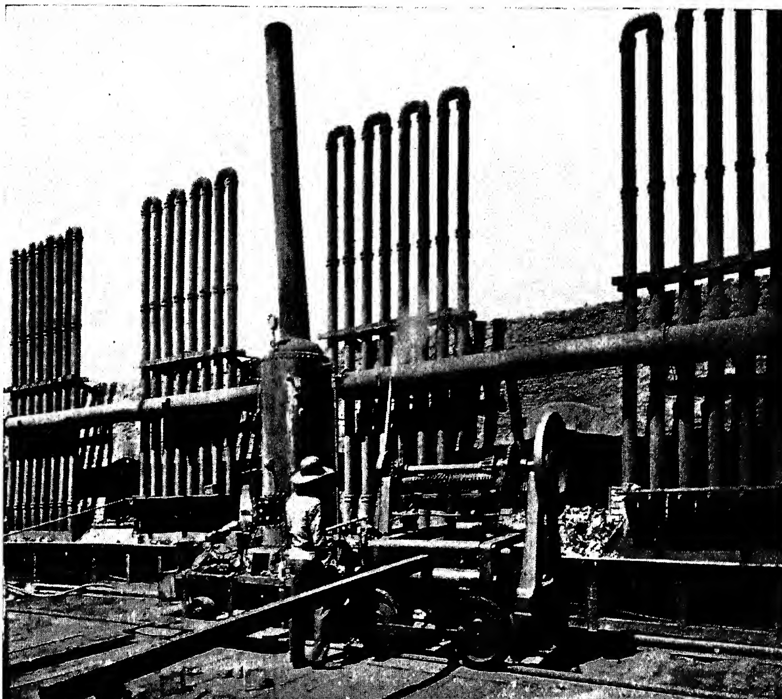


Latrobe Coke-ovens.

naphtha, burning-fluid, heavy-oil (used for lubricating), some hitherto unknown phenols, and, finally, a low percentage of very fine pitch. Carbolic acid has not been found, showing a complete variance in the process from that of the retort system. The great value of the new phenols for medical and antiseptic purposes has rapidly developed, in that respect, a large and important business.

On a test of nine months the by-products recovered were, in addition to the gas, $5\frac{1}{2}$ pounds of ammonium sulphate and 9 gallons of crude oil per ton of coal. The low return of ammonium sulphate was due to the omission of any scrubbing apparatus. Subsequent tests with the use of a scrubber showed the ammonium-yield to be about trebled.

FIG. 5.



Mechanical Coke-Drawer.

The whole plant is of the simplest design, the result of many years of experimentation, and within the comprehension of the ordinary laborers usually employed about coke-works.

The coke cannot be distinguished, either in appearance or in practical use, from that made in bee-hive ovens in the ordinary way.

In burning under this system, the doors are kept closed and luted, only air enough being admitted for the combustion of

an amount of gas sufficient for the coking, thus leaving the coke quite free from any ash; and, on opening the ovens, the charge is found at the same height in front as behind. The yield has been very close to the amount of fixed carbon plus ash, or 70 per cent. in a coal containing about 30 per cent. of volatile matter.

An experimental plant has been put up at Latrobe, Pa., consisting of a block of 30 12-foot ovens in double row, 15 in a row, and staggered.

The chief difference between this plant and the English one first described is that here we pass all the gas through 2 40-foot scrubbers, using a small Sturtevant fan in lieu of chimney-draft.

Figs. 2, 3, 4 and 5 show different views of this plant, and bring out quite clearly the salient features.

It is equipped with longitudinal conveyers on either side, discharging at one end upon a cross-conveyer (shown in Figs. 3 and 4), which, in turn, carries the coke to a sufficient elevation to permit it to be screened into the cars.

The operation of drawing the contents of an oven upon the conveyer by means of the "mechanical coke-drawer" is shown in Fig. 5.

This plant has also been furnished with a simple and extremely efficient apparatus for concentrating the ammoniacal liquor to a standard strength before shipping.

The oil, after separation from the ammoniacal liquor, is drawn into tank-cars and shipped. It possesses a value somewhat in excess of that of the tar-products of the retort-system, and is more easily manipulated.

This plant is, while small, very complete. It is, unfortunately, at the present time, not in operation, by reason of the lack of orders for coke; but with the early resumption of operation now looked for, it will offer a very advantageous opportunity for examination to those whose interest may have been aroused.

A Mechanical Coke-Drawer.

BY ROBERT A. COOK, NEW BRUNSWICK, N. J.

(Pittsburgh Meeting, February, 1896.)

AMONG the new devices for cheapening the cost of material entering into the manufacture of iron is the mechanical drawing of coke, by which the coke from bee-hive ovens is extracted with a minimum of manual labor.

About twenty years ago, an ordinary plate-conveyer was put in use at the plant of bee-hive coke-ovens owned by the firm of Bell Bros. & Co., near Middlesboro, England. This conveyer ran close along the front of the ovens, and carried the coke to the cars. The ovens were drawn by hand, and a very great saving in cost was secured, as one man could draw and clean out twice as many ovens after the conveyer was put in use—he having no forking to do, and the coke being carried away as fast as it was drawn to the mouth of the oven.

The next step was further to eliminate hand-labor as far as possible; and Mr. Thomas Smith of the Thorncliffe Iron Works, near Sheffield, patented in 1891 the machine for drawing coke which I here describe.

This machine, the outcome of several years of experimenting, has now been in successful use since 1891, and there are between twenty and thirty of them in operation at present, and others building.

The machine, as may be seen from Figs. 1 and 2, consists of a wedge-shaped shovel of proper width and thickness, connected with a steel rack of sufficient length to reach the back of the oven. The rack is supported by rolls, and guided by a device operated by a hand-wheel, which directs the shovel in the oven.

The rack is driven by a pinion on a vertical shaft carrying a bevel-gear on the upper end. This bevel-gear is driven by another on the engine-shaft.

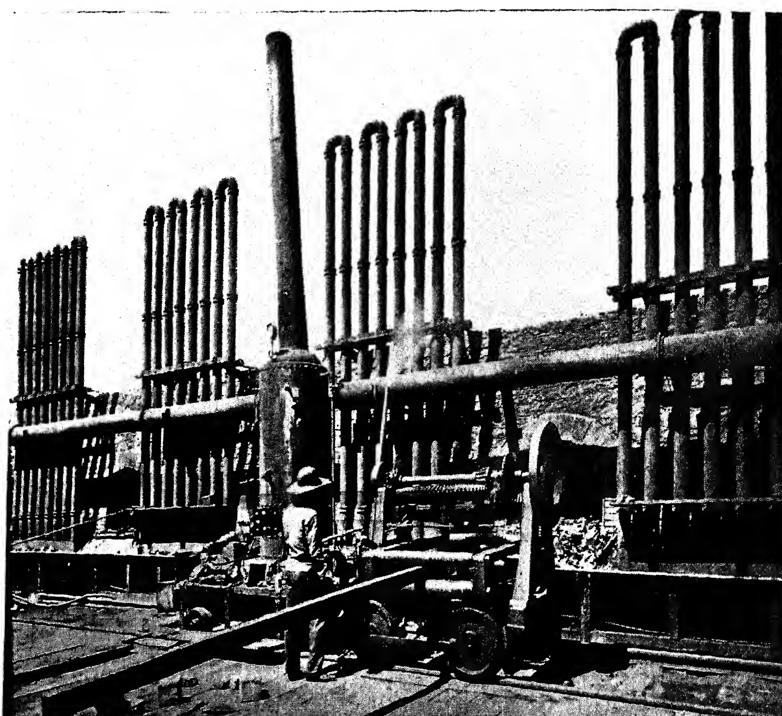
The engine is single-cylinder, with reversible gear; and engine and machine are mounted on a truck, which is propelled

along the track in front of the oven by a worm and gear, connected by a clutch to the engine-shaft.

A nine H. P. boiler, on a separate truck, is coupled to the machine and gives ample steam both for propelling and operating.

In entering the oven, the shovel runs under the coke, and as the shovel is withdrawn, the coke, dropping behind the thick end of the wedge, comes with it.

FIG. 1.

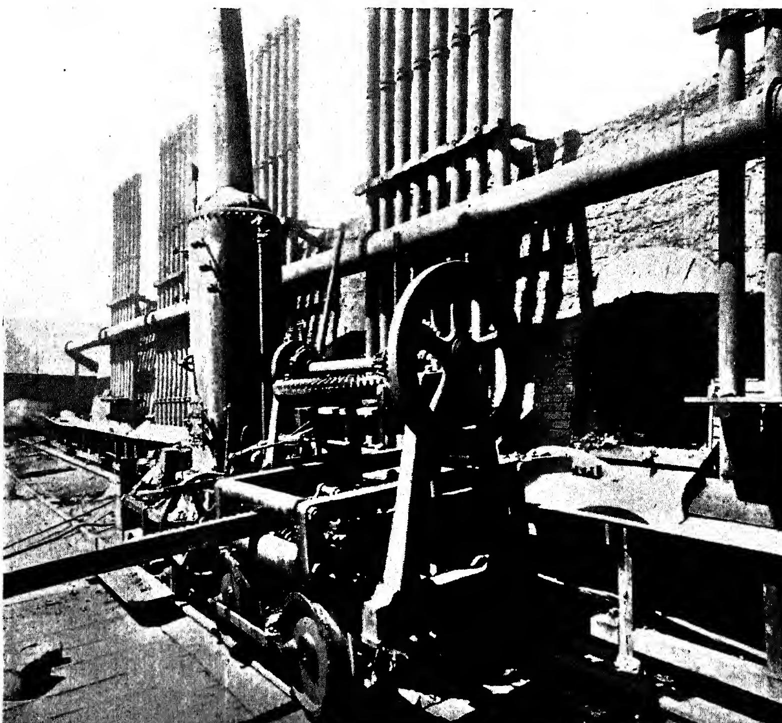


Mechanical Coke-Drawer.

The first of these machines to work in this country was made from English drawings by Messrs. McLanahan & Stone, Hollidaysburg, Pa., and has been in operation for the past month at the plant of the Latrobe Coal Co., Latrobe, Pa. There is also at the same plant an English-built machine, imported from the Thorncliffe Iron-Works, which has not yet been put in operation.

The Latrobe plant, consisting of a block of thirty bee-hive ovens, is just finished and has been in operation for scarcely a month. It cannot furnish a full day's work for one drawer, as there are only fifteen ovens to be drawn daily, instead of from thirty to thirty-five, which is within the capacity of the machine.

Fig. 2.



Mechanical Coke-Drawer.

A green hand on the extractor is now drawing four 12-foot ovens per hour.

Of course, the machine can not get around the jambs, and take out the last of the coke. This is done by hand, and takes from five to ten minutes per oven.

In England, one boy operates the extractor; one man waters; one man levels the ovens and lutes up; and one man rakes out after the extractor, for a plant of sixty ovens.

As the ovens of the Latrobe plant are built in double rows, parallel to the railroad, there are two lines of conveyers running to the end of the block, where a cross-conveyer running at right angles to the other two, rising at an angle of 1 to 4, takes the coke from them and elevates it to a chute, from which it is drawn into the car, after passing over a screen to remove the braize.

To the question, what effect this mechanical work produces on the coke, I can answer that the extractor does not break it any more than, if as much as, hand-drawing; but there is some breakage in the screen; and in looking at a car of screened coke, the fine is seen on the top while the coarse pieces run to the sides.

The coke, once on the conveyer, does not move until it reaches the end and drops on to the cross-conveyer.

These conveyers on each side of the block are 238 feet long, and the end-conveyer is 110 feet. They travel at a rate of 50 feet per minute and take twelve H. P. to run them when loaded. As regards the expense of operating, the same number of men will operate the extractor and conveyer and draw the ovens, but the cost will vary with the wages of different localities. The following men are required: One man at the conveyer-engine (unless it were run by central power-plant), who could operate several conveyers and attend to oiling, etc.; one man to water the coke; one man to run the extractor; one man to rake out; and one man to lute up and level. This number of men is employed at several plants now operating in England.

These extractors can be used without the conveyer, if desired; and the changes to be made in old ovens are simply to widen the oven-doors and then to lay a track in front of the ovens.

It may be of interest to know what wages are paid for drawing ovens in England in the Yorkshire district. For hand-drawing they pay 45 cents per oven. This includes watering, levelling, drawing and loading on cars.

The *per diem* wages for drawing with the extractor are, $1 for engineer, 62½ cents for the extractor-runner and 75 cents for the man who waters; 5 cents per oven is paid for cleaning out after the extractor and 7 cents for levelling, or a total of 19.9 cents per oven, against 45 cents for hand-drawing.

The Magnetic Separation of Non-Magnetic Material.

BY H. A. J. WILKENS AND H. B. C. NITZE, SOUTH BETHLEHEM, PA.

(Pittsburgh Meeting, February, 1896.)

INTRODUCTORY.

AT the Atlanta meeting of the Institute in October, 1895, some brief remarks were made by Mr. Wilkens on the above subject. It is the object of this paper to set forth the substance of these previous statements, embodying therewith additional information gained since that time, and a brief illustrated description of the method and means of the separation.*

Lack of time will unfortunately not allow as complete a *résumé* of the subject in all its phases, especially those relating to the scientific side of the question, as its novelty and importance would warrant, so that even this paper must be considered a preliminary one. It is hoped that at some future meeting of the Institute we may find an opportunity of presenting supplementary data of interest.

The paradoxical title above used requires some explanation. The term "non-magnetic material" is applied to such substances as have heretofore been considered non-magnetic in the arts, more especially in the practice of ore-concentration.

Scientifically, it has been shown by Faraday, Plücker, Wiedemann and other physicists that magnetism is an inherent property common to all substances, solid, liquid and gaseous, *i.e.*, all substances are either attracted or repelled by the poles of a magnet, although in the great majority of cases the manifestation of this

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* NOTE BY THE SECRETARY.—In connection with the oral presentation of this paper by Mr. Wilkens at the Pittsburgh meeting, numerous small samples were exhibited, showing the various products of separation by this method as obtained in experiments at South Bethlehem, Pa.—R. W. R.

force is so feeble as to be measured only by the most delicately arranged physical apparatus.

The substances attracted are known as paramagnetic, and those repelled, constituting by far the greater class, as diamagnetic.

The diamagnetics show but a slight deviation in intensity as compared with air, which is the neutral substance. Bismuth is the most diamagnetic body known, and its diamagnetic permeability, expressed in figures, is stated to be 0.99982, taking air as unity.

The paramagnetics, on the other hand, show a wide variation in intensity or permeability to magnetic forces.

There exists, however, a distinct and wide gap between such as are highly magnetic and such as are only feebly so, and accordingly two classes might be made, namely, such as can be and such as cannot be lifted by the common hand magnet or equivalent power.

To the first class belong the metals iron, nickel and cobalt, and the minerals magnetite and pyrrhotite, and these are the only substances that are generally recognized as magnetic; while all other paramagnetics, comprising the second class possessing comparatively infinitesimal magnetic susceptibility, are regarded in the arts as practically non-magnetic. The abrupt gap between these two classes is not occupied, so far as our present knowledge goes, by substances of intermediate and graduated magnetic susceptibility.

It is with this second class of paramagnetic substances that we are concerned.

Faraday's list, published in 1846, includes a large number of metals, chemical salts and organic substances. It will suffice here to give the list of metals.

Paramagnetic: Iron, nickel, cobalt, manganese, chromium, cerium, titanium, palladium, platinum, osmium.

Diamagnetic: Bismuth, antimony, zinc, tin, cadmium, sodium, mercury, lead, silver, copper, gold, arsenic, uranium,* rhodium, iridium, tungsten.

Various attempts have been made by physical investigators in Europe, with more or less success and reliability of results, to determine the specific magnetic permeability of substances.

* According to later observations of M. Verdet, uranium is paramagnetic.

To give at least some idea of the great gap existing between the commonly known magnetic and some of the other para-magnetic bodies, the following results obtained by Delesse may be cited: Taking the magnetic attractability of steel at 100,000, he found that of magnetite to be as high as 65,000, of siderite only 120, of hematite 93 to 43 and of limonite 72 to 43.

As to the theories concerning the magnetic properties of matter, the works of Poisson, Ampère, Coulomb, Becquerel, Weber, Kohlrausch, Plücker, Tyndall, Faraday, Wiedemann and others would furnish material sufficient for a lengthy paper in itself, and, although the theme is of great interest in this connection, we must confine ourselves to the statement of a few of its more important general features.

The latest accepted theories account for the magnetism of bodies by the existence of magnetic molecules, and explain the variation of magnetic susceptibility as due to the arrangement or structure of these molecules.

The ratio of the sum of the volumes of the magnetic molecules to the entire volume of a body is the magnetic density of that body. It is upon this that the capacity of a body for magnetism seems to depend, and it has been shown to vary with the temperature and other physical and chemical conditions.

It does not appear that the magnetism necessarily depends upon the amount of iron in a substance, or upon the chemical condition of that iron in a ferrous state, as has been thought by many.

Among the most interesting researches are perhaps those of Wiedemann, and some of his conclusions may be mentioned here. He found that:

1. The magnetism of a solution is the direct addition of the magnetism of the solvent and the salt dissolved, and the magnetism of the latter is proportional to the weight contained in the volume.

2. The magnetism of the dissolved salt is practically independent of the nature of the solvent.

3. The magnetism of salts in solution decreases proportionally with the temperature.

4. The magnetism of salts in a dry state is practically the same as that of the dissolved salts, excepting in case of a considerable change in density; and the addition of water of crystal-

lization to a salt which is naturally free from water, does not materially alter the magnetism.

5. The specific magnetism of a salt is the temporary magnetic moment induced by a magnetizing force of unity in the unit weight of the salt, and in salts of analogous composition of the same metal the product of the specific magnetism with the atomic weight is constant. (Identical with the law of specific heat.)

6. The atomic magnetism of similarly constituted oxygen and haloid salts of magnetic metals is approximately equal; and this applies to the oxyhydrates also.

7. The atomic magnetism of magnetic metals is equal in the exchange from one binary compound to another similarly constituted one, and it is equal to the sum of the atomic magnetisms of the constituents.

8. Colloidal iron oxide has a magnetism of only 21 per cent. of that of iron oxide and oxyhydrate; therefore the magnetic atom-grouping shows essentially different properties.

9. The sulphides of the magnetic metals are usually very much less magnetic than the corresponding salts, and this must be due to a different grouping of the magnetic atoms.

10. Two simple diamagnetic elements (like copper and bromine) may form a magnetic compound, and a magnetic element in combination with an indifferent or weakly diamagnetic one may lose its magnetism.

The knowledge of the paramagnetic property of many substances has been of great scientific interest since the time of Faraday's demonstration in 1845. In 1871 Fouqué, and since then Doelter and other petrographers, have made use of this knowledge to some extent in the mineralogical laboratory. But it has never gone beyond this point, *i.e.*, to an economical and practical application in the useful arts. Practical engineers and inventors have either taken for granted that the magnetic susceptibility of substances was constant, and about what was indicated by the ordinary hand magnet, or, if they were acquainted with the universal magnetic permeability of substances, they have felt that the attainment of sufficiently high power to affect the extremely low susceptibility of the paramagnetics, other than iron, nickel, cobalt, magnetite and pyrrhotite, was impracticable within the bounds of practical economy. At all events,

they have been discouraged from making any attempt to design electro-magnets capable of such peculiarly concentrated and condensed a field as to attract these substances.

Even in the present case useful results were achieved only after making many experiments and actual trials carried on, in the beginning, without any knowledge of the previous theoretical investigations.

A large number of machines have been invented and perfected for concentrating the so-called magnetic minerals, that is, magnetite and pyrrhotite, chiefly the former. When it became desirable to separate other ferruginous minerals, such as hematite, limonite, siderite, etc., which were, in the generally accepted sense, non-magnetic, methods for converting them into artificial magnetite were resorted to. This was done either by an oxidizing or a reducing-roasting in or without the presence of carbon-contact.

As examples may be mentioned the treatment of limonite ores as suggested by Clemens Jones* and others; the treatment of the red fossil hematite ore of the Birmingham district in Alabama, described by Dr. W. B. Phillips;† the practice of the Wythe Lead and Zinc Company, Austinville, Va., for separating limonite from calamine;‡ and the separation, by this means, in many European localities, of siderite from zinc-blende, as at the Friederichssegen§ and the Ludwigseck‖ mines, in Germany; Przibram¶ and Maiern,** in Austria; Allevard,†† in France; the Mercadal mines,‡‡ in Spain; Monteponi,§§ in Sardinia, etc.

The reduction of iron and nickeliferous pyrites to the magnetic subsulphide, and of nickel oxide to metallic nickel, has been suggested by Eustis and Howe among others,‖‖ but experi-

* *Trans*, xix., 289, Sept., 1890.

† *Trans.*, xxv., 399, Oct., 1895.

‡ *Jour. Iron and Steel Inst.*, London, 1894, vol. xlv., p. 428.

§ M. Bellom. *Annales des Mines*, Ser. 8, vol. xx., 1891, pp. 5–186.

‖ *Preuss. Zeitsch.*, vol. 41, B, p. 208.

¶ *Trans.*, ix., 451. Ellis Clark, 1881.

** *Oesterr. Zeitsch.*, 1893, Nos. 4 and 5.

†† G. Gromier. *Bull. de la Soc. de l'Indust. Min.*, vol. 7, p. 465.

‡‡ G. Prus. *Le Génie Civil*, vol 17, p. 337.

§§ *Oesterr. Zeitsch.*, 1892, No. 20.

‖‖ *Trans.*, x., 305, 1882.

ment has shown this to be a very delicate operation, and it has never been put into practice on a working-scale.

On the same general principle, Mr. G. G. Convers designed in 1892 a process for the separation of the franklinite-ore of the Franklin mines, New Jersey. This ore is a crystalline granular aggregate of franklinite, willemite and calcite, with smaller amounts of zincite, garnet, fowlerite, tephroite, etc. It was the object to obtain the willemite and zincite as free as possible from franklinite (the principal iron- and manganese-bearing mineral), so that the former could be charged directly into the retorts of the Belgian furnaces for the manufacture of spelter—to which use the presence of iron or manganese, attacking the material of the retorts, would be a fatal objection. Mr. Convers roasted the ore, mixed with fine anthracite coal, in a revolving cylinder-furnace, thus obtaining a direct reducing-action and rendering the franklinite magnetic, so that it could be separated on Wenström magnetic drums. This process gave good results; but the cost of roasting and the uncertainty of a uniformly magnetic product led to further experimenting, and finally the method, a direct magnetic concentration of these ores, without previous roasting, was evolved by Mr. J. P. Wetherill, manager of the Lehigh Zinc and Iron Co., South Bethlehem, Pa.

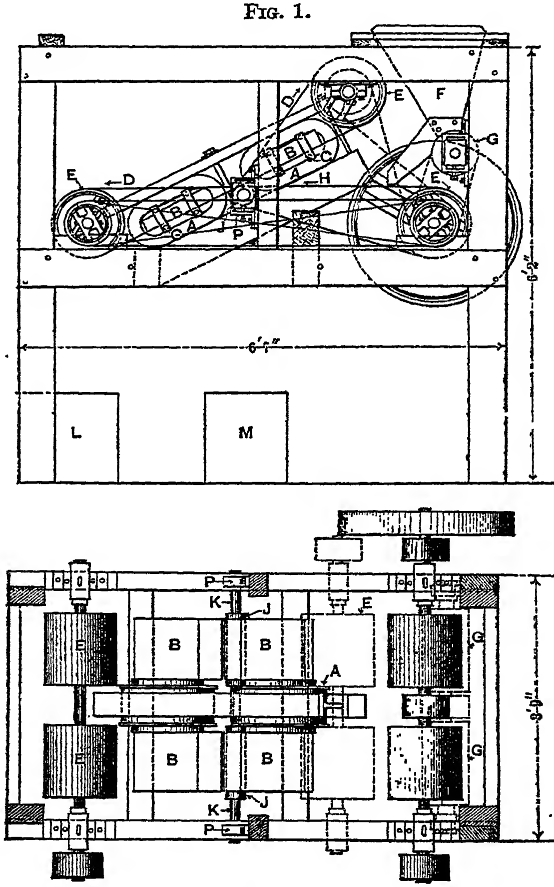
Mr. Wetherill found that not only the franklinite, but also the garnet, fowlerite, tephroite and similar deleterious iron- and manganese-bearing minerals could be thus eliminated.

These surprising and important results were attained by utilizing a peculiarly high magnetic power, applied in machines especially adapted for the purpose, and sufficient to act upon the comparatively almost infinitesimal magnetic permeability of these minerals, instead of attempting (as had heretofore been done) to raise this low permeability to a point known in practice as "magnetic," that is, capable of attraction by a common hand-magnet or the electro-magnets heretofore employed.

Further investigations showed that a large number of ferruginous and manganiferous minerals, such as hematite, limonite, siderite, garnet, pyrolusite, etc., besides many chemical salts of iron, manganese and chromium were affected by this extraordinary magnetic power, and were thus adaptable to this method of concentration.

Letters-patent were granted to Mr. Wetherill for his method of direct magnetic separation of paramagnetic substances of low permeability from mixtures containing the same, as well as for the machines employed to obtain this result; and these patents are now owned by the Wetherill Concentrating Co., located at South Bethlehem, Pa.*

FIG. 1.



Wetherill Magnetic Separator, No. 1.

METHOD AND APPARATUS.

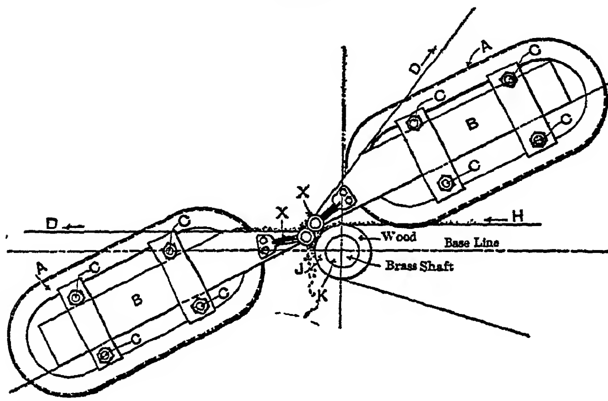
The machines for making these separations are of two types. Figs. 1 and 2 show a machine more especially adapted for

* U. S. patents Nos. 555,792 (dated March 3, 1896), and 555,794 (dated March 3, 1896).

the treatment of fine ores, and for concentrations in which rich magnetic heads are more particularly sought. It has been operated very successfully on the Clinton fossil-ores of the Birmingham district, Alabama.

In this type the magnetic particles are lifted away from the original mixture. The machine consists of two magnetic cores and bobbins A and four pole-pieces B, which are pointed in

FIG. 2.



Enlarged Section from Fig. 1, showing Arrangement of Pole-pieces and Feed-belt.

the manner shown and are adjustable by the bolts C, so that they may be moved nearer or farther apart as desired.

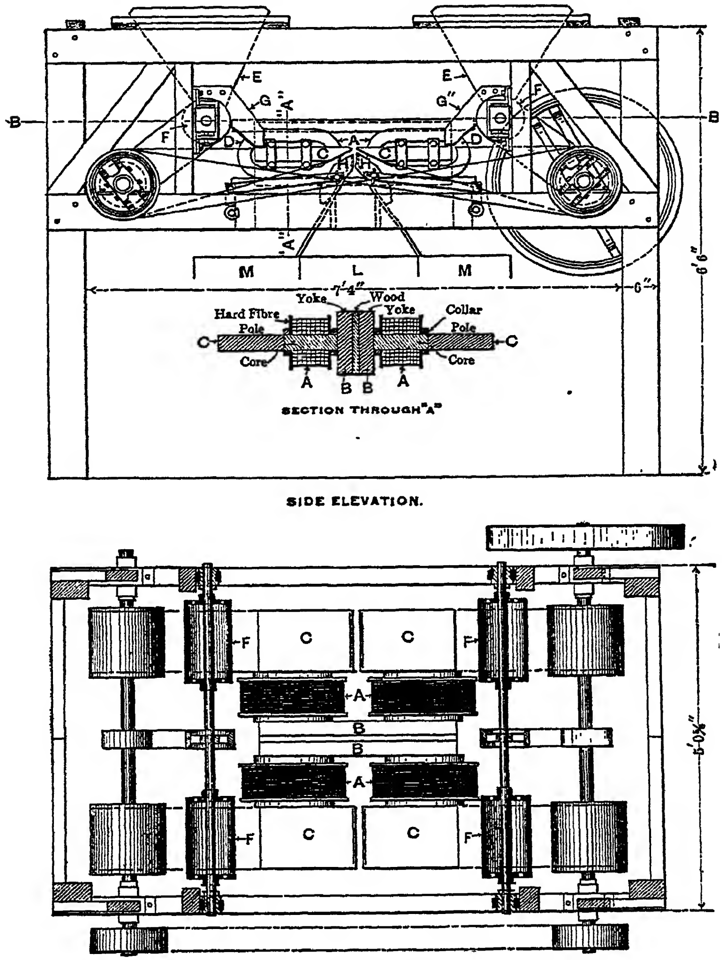
For substances of very low magnetic permeability two of the pole-pieces are dispensed with, and a solid yoke is used instead. For substances of higher magnetic permeability, such as garnet, franklinite, siderite, etc., the yoke may be removed and two pole-pieces substituted for it in the manner shown. About each pole-piece revolves a belt D (driven by a pulley E) in the direction shown by the arrows.

The ore is fed from a hopper F by means of a feed roller G, upon a belt H, which carries it in a thin layer, say $\frac{1}{8}$ to $\frac{3}{16}$-inch thick, to and about the pulley J, which is of small diameter and is upon a brass axle K, which may be raised or lowered by the adjustable bearing P. The feed can be accurately adjusted by means of a sliding shutter at the discharge of the hopper F.

The ore is thus delivered in close proximity to the space between the poles, and the magnetic particles are withdrawn and

lifted up into the highly intensified field existing at this point. They are removed by the horizontal belt D, and carried into a receptacle L.

FIG. 3.



Wetherill Magnetic Separator, No. 2.

The non-magnetic tails fall from the belt H into the receptacle M.

The intensity of the magnetic attraction can be accurately adjusted by changing the distance of the feed-belt from the

pole-points; or by changing the distance between the pole-pieces, or, finally, by changing the ampèrage of the current.

Fig. 3 shows a machine adapted more especially for coarser sizes; material as large as ½-inch in diameter has been handled with it. Excellent results have been obtained by its use on franklinite ores, where clean non-magnetic tails, consisting of willemite and zincite, are sought.

In this type of machine the magnetic particles adhere to the feed-belt as it rounds the pole-points, while the non-magnetic material drops away from it. The apparatus consists of magnets with cores and bobbins A, yokes B, pointed pole-pieces C, belts D, revolving about the pole-pieces in the manner shown, ore-hoppers E, feed-rollers F, and shutes G, by means of which the ore is fed in a thin layer upon the belts D. These belts travel in the direction shown by the pointed arrows and the ore is delivered directly into the opening between the two pointed pole-pieces; means are provided by which the pole-pieces may be set further apart or closer together, as desired. Two shutters H, one beneath the point of each pole-piece, are adjustable, so that the magnetic particles which adhere slightly to the pole-pieces are carried to one side by the moving belts D, into the receptacles M and M on either side of the shutters, while the non-magnetic fall into the central space between the shutters, and thence into the receptacle L.

The degree of concentration desired can be regulated by changing the position of the shutters and the speed of the feed-belts, or by changing the distance between the pole-points, or by changing the ampèrage of the current through the coils.

So far machines have been used with a capacity of from ¾ to 3 tons per hour, depending upon the number of pole-pieces used, the speed of the belts and the nature and size of the material operated on. By widening the pole-pieces and the feed-belts, the capacity can be raised; but this may ultimately reach a point where it will be more advantageous to multiply the number of machines instead of further increasing their size.

The amount of ampèrage necessary for producing the extremely high field is comparatively small. This is due to the large-sized and correctly proportioned cores, yokes and pole-pieces employed, and the almost perfect condensation at the

pole-points of the vast number of magnetic lines generated in them.

For the separation of the Franklin minerals, where the magnetic heads produced consist of franklinite, tephroite, rhodonite, garnet, etc., from 3 to 8 ampères, with from 16 to 30 volts, are necessary. In the concentration of the Clinton red fossil-ore from 4 to 8 ampères were used. For limonite and pyrolusite a somewhat higher ampèrage is employed, from 10 to 15 giving a good separation.

To illustrate the extreme condensation of magnetic lines at the pole-points, it may be mentioned that the current of one blue-stone cell (such as is used in telegraphy) is too powerful to permit a satisfactory separation of magnetite. For this purpose its strength must be reduced by means of resistance-coils, in order to prevent tangling and allow the belts to draw the ore away from the pole-points.

The ability to adjust delicately the magnetic intensity of these machines permits, in many cases, the practical isolation from one ore of several products showing a slight difference of magnetic permeability. For instance, in the concentration of monazite sand the pure monazite can be separated from garnet and rutile at one operation, and the latter, by a slight change in the intensity of the current, can again be separated from each other.

If two such unlike properties as magnetic permeability and specific gravity can be compared, it might be said that a much smaller difference in the former is necessary to permit a separation by magnetic means than would be required in the latter for making a water- or air-separation.

It would be difficult, however, to tabulate the minerals in distinct groups capable of being magnetically separated from each other, for the reason that the magnetic permeability of each mineral, *per se*, is subject to wide variations.

Aside from the separation of several minerals from one ore, the discriminating power can be utilized for the separation of a desired quantity or quality of heads, where the magnetic mineral varies in itself in permeability. For example, in the concentration of the Clinton fossil-ores, to be mentioned hereafter, the magnetic permeability of the hematite particles increases apparently with their percentage of iron, and either a small amount of high-grade heads can be produced with a low

ampèrage, or a large amount of low-grade heads with a higher one. From a commercial standpoint, the exact quantity or quality which it would be advisable to produce would depend upon the increase in value of the concentrates per unit.

The machines above described and illustrated are adapted to dry concentration. Their arrangement for wet concentration would necessitate but a slight change in some of the mechanical features.

Applications.

Among the minerals which have been found susceptible of attraction by the condensed magnetic power of these machines we may mention red hematite, brown hematite, siderite, chromite, menaccanite, rutile, franklinite, pyrolusite, psilomelane, tephroite, rhodonite, garnet, and, in fact, almost all the minerals containing iron or manganese or both; likewise the salts of iron, manganese and chromium.

It is a curious fact that the manganese salts are more magnetic than the corresponding iron salts; for instance, manganese sulphate required 1 ampère, while ferrous sulphate required 8 ampères to be attracted. This agrees with the fact that the manganese silicates, tephroite and rhodonite, show a very high degree of magnetic permeability.

The only iron-bearing mineral which does not come within the attracting limit of the magnets so far constructed is iron pyrites. Higher-power magnets are now being built, with which it is hoped to handle this material successfully.

It is difficult as yet to judge of the full extent of the field to which this method of concentration is applicable. Its original application to the franklinite ores has been mentioned above. It should supplant those operations already enumerated, in which a magnetizing roasting has heretofore been found necessary before magnetic concentration, not only on account of the additional cost which the latter entails, but also because better results may be expected from the direct treatment of the raw material. Magnetizing-roasting is attended, in most cases, with considerable difficulties, such as the sintering and slagging of the ore, imperfect magnetization, etc., which not only decrease the quantity, but may also impair the quality, of the resulting concentrates. Such difficulties were encountered in the

magnetization of the red fossil-ore of the Birmingham district and in the roasting of siderite-blende ores in several European localities.

It will prove the saving process for the concentration of certain ores and mineral combinations which are not capable of separation by gravity-methods, as, for instance, in the separation of the constituents of monazite sand (rutile, garnet and monazite), the minerals of the franklinite-ore, and many of the iron-ores.

Perhaps its widest application will be in the field of concentrating iron-ores, red and brown hematites and carbonates. It is estimated that over 90 per cent. of the material charged into the blast-furnaces in this country is of these classes, and that in European countries the proportion is even larger. These ores will practically be placed on the same level with magnetite, so far as the ability to concentrate them magnetically in the raw state is concerned.

Of importance to the iron and steel industry will be the concentration of manganese-ores, with which other methods of concentration, such as jigging, have always experienced difficulty and waste.

There are no doubt many other special cases in which this process of direct magnetic concentration may become applicable.

Among the materials, besides those mentioned above (pp. 356, 358 and 361), which have been treated successfully are:

1. A mixture of apatite and rutile, occurring in a deposit near Charlottesville, Va.

2. Monazite-sands from North and South Carolina.

3. Garnetiferous rocks and schists, from which it was desired to extract the pure garnet.

4. Corundum ores, from which garnet and other deleterious ferruginous minerals were to be removed.

5. Ores containing siderite as a deleterious ingredient, as in cryolite, for instance.

In certain metallurgical operations this method of magnetic concentration may form an intermediate stage, as, for instance, in the separation of pyrite from blende, after the mixture has received a dead roast to prepare the blende for the retorts.

Its application has not been exploited as yet in the chemical and other industrial arts.

Franklinite-Ores—The process has been in operation on a large experimental scale at the works of the Lehigh Zinc and Iron Co., South Bethlehem, Pa., for the separation of the franklinite-ores. Some 15,000 tons of material have been treated here, and, on the basis of the experience thus gained, a large and comprehensive plant has been designed and erected at the mines of the Sterling Iron and Zinc Co., Franklin, N. J. This plant, having a capacity of 400 tons of raw ore per day, is about ready to go into operation. The ore, as it comes from the mine, is dried, crushed to 10-mesh size by means of Blake crushers and rolls, and sized. The franklinite, tephroite, rhodonite, garnet, etc., are removed on the Wetherill machines, leaving, as non-magnetic tails, a mixture of willemite, zincite and calcite, from which the latter is subsequently separated by jigging. The operation of the process on this material is highly successful, as is shown by the following analyses:

	Weight. Per cent.	ZnO. Per cent.	Fe. Per cent.	Mn. Per cent.
Run-of-mine ore, ...	100	31.09	20.34	9.34
Franklinite separated, . . .	56	24.38	35.50	13.62
Willemite and zincite separated, .	28	60.00	1.50	6.10*
Calcite separated, ...	16	4.00	......	......

Clinton Fossil-Ores.—Of more general interest, on account of the greater application of the process and the large extent of the field, are perhaps the results obtained on the red fossil hematite-ores of the Birmingham district in Alabama.

The richer, soft ores of this district, such as are used in the furnaces, average from 45 to 48 per cent. in iron and from 30 to 24 per cent. in insoluble matter. Such ores occur, however, only in few localities, which are limited in extent, and are now almost exhausted. By far the greater portion of the leached ore-beds consists of material running from 35 to 45 per cent. in iron and from 45 to 30 per cent. in insoluble matter.

This latter class of ore cannot be used in the furnaces to advantage, and is therefore practically worthless, unless the percentage of iron be raised by concentration, and at the same time the insoluble matter be proportionately decreased.

Structurally, the ores are as a rule fine-grained, the average

* This manganese is to a great extent chemically contained in the willemite and zincite.

size of the distinct particles being such as would pass through a 10-mesh screen. On examining the product of separation it is seen that the ore consists of:

1. Rounded silica grains, which, owing to a coating of iron oxide, are found by analysis to contain from 10 to 15 per cent. of iron.

2. Rounded grains of more highly ferruginous material, running perhaps 30 per cent. in iron.

3. A binding-material of hematite, which in itself carries a varying amount of insoluble matter, depending upon the locality of the ore, fineness of grain, etc.

It was found that a mere elimination of the sand-grains was not sufficient to raise the heads to the desired quality, but that the material of the second group, and in some cases a portion of the more siliceous matrix, had to be separated and classed either with the tails or as middlings, in order to obtain high-grade heads.

The tests on these ores were made under the auspices of the Tennessee Coal, Iron and Railroad Company, and were in direct charge of Dr. William B. Phillips, the consulting chemist of the company, who, for the past three years, has made a specialty of the problems involved in the concentration of the Birmingham ores. The plant used was the one at which Dr. Phillips had carried on his experimental work in magnetizing-roasting and concentration, and which was described in his paper on that subject.* A small inclined dryer was substituted for the magnetizing-kiln, for drying such ores as carried a large amount of moisture. It was found that the ore stored in the stock-house could generally be prepared and treated without drying. A revolving screen, which divided the material into three sizes, (through 8- on 15-, through 15- on 40-, and through 40-mesh) was added. Two Wetherill machines, one of each type, were used for making the separations.

Various working-tests were made on material from a great number of localities, and the results were verified by some 500 analyses made in the laboratory of the company.

Space will not permit of a detailed account and discussion of these results; it is merely intended here to present a general

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* *Trans.*, xxv., 399.

idea of what was accomplished. The previous magnetization-experiments had been made entirely on the richer soft ores, such as are now being used directly in the furnace, and of the composition given above.

Concentration-tests on this material by the Wetherill process gave the following results (calculated on a basis of 100 tons of raw ore):

	Iron. Per cent.	Insoluble. Per cent.
Original ore,	48.03	25.20

	Iron. Per cent.	Insoluble. Per cent.
57 tons of heads, containing	57.10	13.10
28 tons of middlings, containing	46.20	25.40
15 tons of tails, containing	10.00	70.80

It was further found that about 20 per cent. in weight of this raw ore could be brought up to: iron, 59.15; insoluble, 10.45 per cent.

At the prices offered for concentrates by the T. C. I. & R.R. Co., it would be perfectly feasible to purchase this grade of raw ore at the market-price, and make the concentrates at a profit. As, however, the ore of this quality is almost exhausted, the question of concentrating it on a large scale will hardly arise.

The above results compare most favorably with those previously obtained by the magnetizing-roasting process, particularly in the proportional amount of heads that were produced and the comparatively small percentage of iron carried in the tails.

For the purpose of comparison, the following results of that process (calculated on a basis of 100 tons of raw ore) are given:

	Iron. Per cent.	Insoluble. Per cent.
Original magnetized ore,	49.05	22.05

	Iron. Per cent.	Insoluble. Per cent.
15 tons of heads, containing	59.00	11.06
35 tons of middlings, containing	52.00	20.00
50 tons of tails, containing	44.00	28.00

Only the more perfectly magnetized material was used on the concentrating machines.

In the magnetizing process is to be considered not only the cost of roasting, but also the imperfections attending it, such

as the incomplete magnetization, the louping of the ore-lumps and the inability to use a large percentage of fines in the kiln.

These difficulties may, of course, be overcome to some extent with more experience in practice, and in this direction Dr. Phillips and his assistants have shown great ingenuity.

There is no doubt, however, that the raw material is better adapted for concentration, on account of the uniformity in the magnetic properties and physical structure of the several ingredients.

The tests by direct concentration on the lower-grade ores showed a proportionately greater increase in the percentage of iron than those on the higher-grade material. The quality of the heads was, however, not as good, which shows that the hematite matrix in the low-grade ores carries a larger percentage of inherent insoluble matter than that of the richer ores.

Among others, the following results were obtained (calculated on the basis of 100 tons of raw ore):

	Iron. Per cent.	Insoluble. Per cent.
Original ore,	41.58	37.51
	Iron. Per cent.	Insoluble. Per cent.
69 tons of heads, containing	52.00	23.00
31 tons of tails, containing,	18.40	70 00

About 25 per cent. in weight of this original ore was raised to: iron, 56.40 per cent.; insoluble, 17 per cent.

Tests were also made on the so-called "hard ore," which represents that portion of the ore-bed from which the lime has not been leached. The raw ore of this character, as used at the furnaces, averages: iron, 35.50; insoluble, 17.50; lime, 16 per cent. From this were obtained from 50 to 60 per cent. in weight of heads, containing: iron, 48; insoluble, 10.50; lime, 10 per cent.

The cost of mining this hard ore, on account of its being underground, is far in excess of that of the soft ore. And, therefore, although the use of such concentrates as were produced would be of great advantage in the furnace, the cost of mining 2 tons of raw ore would be too great to permit the use of 1 ton of concentrates obtained therefrom at a profit under present circumstances.

The above tests must be considered to a certain extent preliminary. There is no doubt that a wider experience with this material, and, as a rule, a somewhat finer crushing and more judicious sizing, will give still better results. Experiments have gone far enough, however, to prove that this process of direct magnetic concentration is not only technically the most feasible, but also commercially the most economical one.*

Brown Hematite-Ores.—To show the application of the process to brown hematite-ores, the following results, from experiments made at the testing-plant of the Wetherill Concentrating Co., South Bethlehem, Pa., may be of interest:

Brown Hematite from Iron Gate, Va., Alleghany Iron Co.

	Percentage by weight.	SiO_2. Per Cent.	Fe. Per Cent.
Original ore,	100	31.29	43.08
Concentrates,	63.4	11.24	51.04
Tails,	36.6	66.00	31.74

Brown Hematite (Washer-Tailings), Barren Springs, Va.

	Percentage by weight.	SiO_2. Per cent.	Fe. Per cent.
Original Ore,	100	29.93	32.03
Concentrates,	30	7.43	53.14
Tails,	70	39.58	22.98

Jigged Zinc-Ore (Consisting of Limonite and Calamine), Wythe Lead & Zinc Co., Austinville, Va.

	Percentage by weight.	Fe. Per cent.	Zn. Per cent.
Original Ore,	100	18.60	29.57
Concentrates,	33	49.45	5.58
Tails,	67	3.41	41.40

Manganese-Ores.—Although several tests have been made on manganese-ores, they do not show to full advantage the application of the process to this material, as the pyrolusite was in itself siliceous through chemically combined silica.

As showing the practicability, however, of concentrating manganese-ores, the following test is given:

* It is regretted that business negotiations and commercial calculations are not far enough advanced to permit figures of costs and value of the concentrates to be published at the present time.

Manganese-Ore (Culls from Waste-Heap, Consisting of Particles of Chert in a Matrix of Siliceous Pyrolusite), Cave Spring, Ga.

	Percentage by weight.	SiO_2. Per cent.	Mn. Per cent.
Original Ore,	100	43.00	28.78
Concentrates,	52	20.85	40.91
Tails,	48	67.20	15.54

Conclusions.

In the application of this process, limitations will be encountered similar to those common to other methods of concentration. The constituents of the material to be concentrated must exist in a visible mechanical mixture, so that, after crushing, each constituent may be present in separate and distinct particles. As already stated, the Wetherill machines have successfully handled material up to ½-inch in diameter, and a coarser crushing than this is seldom advisable in concentration. As a rule, finer crushing is necessary on account of the structural condition of the raw material, as is the case in the Clinton fossil-ores.

The antipathy against the use of fine ores in the blast-furnace has been greatly overcome by the successful utilization of the Mesabi fine ores in the Pittsburgh district, where 33 per cent. of material, running less than 8-mesh in size, constitutes the regular furnace-charge, and as high as 60 per cent. has been used without serious difficulty. Briquetting may, in some cases, be resorted to, especially where a self-fluxing and partially self-reducing briquette can be made, and where these advantages counterbalance the additional cost of manufacture.

When the difference between the magnetic permeability of the constituents is marked, a preliminary sizing of the material is unnecessary; but this must be resorted to, and the number of sizes be proportionately increased, as the magnetic difference becomes less. The process resembles in this respect an air- or water-separation. In rare instances this difference may be practically *nil*, and a separation may become impossible. Such was found to be the case in the attempted separation of chromite and serpentine in a certain chrome-ore, these two ingredients showing practically the same magnetic properties.

Experience has shown that the actual cost of magnetic concentration, under usual conditions, will not exceed five cents

per ton of raw ore. The cost of preparing the ore for the machines by drying, crushing and sizing will depend to a great extent upon the character of the ore, locality of the deposits and other conditions, and will not differ from the corresponding cost under any other method of concentration.

The Actual Accuracy of Chemical Analysis.*

BY F. P. DEWEY, WASHINGTON, D. C.

(Colorado Meeting, September, 1896.)

THE subject of this paper does not embrace the consideration of ways and means for the increase of analytical accuracy, or the question what could or should be attained in that direction. I desire simply to call attention to the degree of accuracy exhibited in actual every-day practice. In estimating this, little weight will be given to the evidence afforded by the agreement of duplicate or multiple determinations by the same chemist; for I am convinced that such agreement is a delusion and a snare. Nor will special importance be attached to the agreement of two or even three analyses in special cases, or to the agreement between two methods practiced by the same analyst. I propose to compare the results obtained by several chemists, working upon the same sample and by various methods, in order to exhibit, as I have said, the actual condition of practice.

The available material for illustrating this phase of the question is unfortunately scanty; but something has been done; and I hope, by calling attention to some of the work in this line, to stimulate further work in the same direction by inducing others to prepare suitable samples and submit them to various chemists who are competent and willing to make the necessary determinations, and fully describe the methods they employ.

I draw most of my illustrations from the *Transactions* of the Institute, the *Proceedings* of the Association of Official Agricultural Chemists, and from personal experience.

Manganese in Steel.—In May, 1881, Mr. William Kent presented a paper to the Institute entitled "Manganese Deter-

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* This paper was, by arrangement, read also before the American Chemical Society.

minations in Steel,"* in which he gave twenty-four determinations of manganese, made by ten different chemists, employing two main methods, on samples from a plate of steel. These results presented the remarkable range of from 1.14 per cent. to 0.303 per cent., and one chemist reported results ranging from 1.14 per cent. to 0.434 per cent.

A portion of this variation was undoubtedly due to variations in the samples, since the same sample was not used throughout by the different chemists.

Throwing out the anomalous result of 1.14 per cent. we have twenty-three determinations running from 0.619 to 0.303, with an average of 0.415 per cent., thus showing that at that time the determination of manganese in steel, when only about 0.4 per cent. was probably present, might exhibit an extreme variation between the highest and lowest results of about 0.3 per cent. or 75 per cent. of the amount of manganese present.

These results were certainly very discouraging; but if they did nothing else, they served to call attention to the very unsatisfactory character of the determination of manganese in steel at that time.

I do not recall any recent symposium on the determination of manganese in this class of material; but in 1886 Captain A. E. Hunt,† in giving a measure of the accuracy of the colorimetric method, speaks of a variation of 0.02 per cent. in steels containing 0.15 to 1.5 per cent. of manganese as "sufficiently accurate for all practical work," thus clearly intimating that the current results of analysis by other methods were at least as good. This degree of accuracy, if attained by different chemists upon the same sample, must be considered a satisfactory advance over the results reported by Mr. Kent.

Early in 1883 Mr. G. C. Stone began a series of contributions on the "Determination of Manganese in Spiegel."‡ In his first paper he reported 13 determinations by 5 chemists, all working upon the same "works"-sample, showing from 15.49 to 13.83 per cent., and also 26 determinations by 10 chemists, all working upon a sample of the same spiegel, prepared with especial care jointly by Mr. Stone and one of the other chemists,

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* *Trans.*, x., 101.

† *Trans.*, xv., 104.

‡ *Trans.*, xi., 323; xii., 295, 514.

showing from 14.56 to 10.36 per cent. But some of the low results were obtained by experimental methods.

In the fall of 1883 Mr. Stone reported 20 additional determinations by 5 other chemists, ranging from 14.20 to 10.76 per cent., the extremes being reported by the same chemist when working by different methods, his favorite method giving from 13.84 to 13.65 per cent.; the three low results (less than 11 per cent.) were obtained by the Williams method. In this connection Mr. Stone presented an interesting table, dividing the methods used into 4 classes and the results into 3 classes, giving respectively, below 13; between 13 and 14; and above 14 per cent.

In the spring of 1884, Mr. Stone reported 27 new results, 19 by 4 new chemists, and 8 by one previously reported, whose new results were obtained by several methods.

We have thus 73 determinations by 19 different chemists. Of these, 2 are thrown out on account of the method used, and 11 "because the chemists were not entirely satisfied with them," leaving 60 determinations by 18 chemists, using 12 methods.

These 60 results range from 14.47 to 12.60 per cent. and average 13.39 per cent. Leaving out 8 determinations by one method which is considered to give low results, the lowest determination becomes 12.92 and the average 13.48 per cent., showing an extreme variation of 1.55 per cent. of manganese between the highest and lowest results, and showing only 44 per cent. of the results within 0.2 per cent. of the average.

In the discussion of Mr. Stone's second paper, Mr. J. B. Mackintosh* presented an analysis of Mr. Stone's first 46 results, retaining the results by the Williams method, from which he argued that the evidence pointed to 12.956 per cent. as the true content of manganese in this spiegel. If this is the case then there is a very decided tendency to get too high results in this class of work.

Taken as a whole, this investigation would seem to show that variations of 0.5 per cent. in the determination of manganese in this grade (10 to 15 per cent. Mn) of spiegel are to be expected, and much wider variations may be found.

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* *Trans.*, xii., 300.

Phosphorus in Pig-Iron.—Early in the 80's, Messrs. Potter and Riggs, of St. Louis, Mo., sent out a sample of pig-iron for the determination of phosphorus.

This examination yielded 26 results, by 11 chemists, using 5 methods, ranging from 0.181 to 0.141 and averaging 0.160 per cent., and showing an extreme variation of 0.040. The maximum variation reported by any one chemist was 0.017 per cent., while three reported duplicates agreeing to 0.001 per cent. These results have never been published. One of the chemists discovered arsenic in the sample, which would account for some of the variation in the series. His determinations, in duplicate, were 0.151 and 0.152 per cent.

In February, 1882, Mr. F. E. Bachman presented a paper to the Institute,* in which he reported 44 results by 18 chemists, using 4 methods, ranging from 0.165 to 0.096 and averaging 0.143 per cent. The extreme variation was 0.069 per cent. The maximum variation reported by any one chemist on straight duplicates was 0.01 and the minimum 0.0004 per cent. Experimental determinations by Mr. Bachman, using different processes, yielded variations amounting to 0.043 per cent.

At the Atlanta meeting, in October, 1895, Mr. George E. Thackray presented a paper entitled, "A Comparison of Recent Phosphorus Determinations in Steel." † He first gives a table of determinations of phosphorus by two chemists on 8 samples, the differences ranging from 0.033 to 0.012 per cent., one chemist uniformly getting high results. One found from 0.080 to 0.074 per cent., and the other 0.110 to 0.088 per cent. in these steels. These results were manifestly unsatisfactory.

A second table shows results by 3 chemists, the buyer's, the seller's and an arbitrator. By the arbitrator's determinations these steels carried from 0.087 to 0.063 per cent. of phosphorus.

The maximum difference in any set of 3 results was 0.017 and the minimum 0.005 per cent.

These results were obtained in the settlement of sales. As a result of the discussion which accompanied the matter, two samples of steel were prepared and sent to various chemists. A fourth table gives 36 results obtained by 23 chemists, using 29 methods on one steel, showing results averaging 0.0496 and

* *Trans.*, x., 322.

† *Trans.*, xxv., 370.

ranging from 0.055 to 0.045 per cent., an extreme variation of only 0.010 per cent. Any individual result was practically within 0.005 per cent. of the average.

On the second sample 38 results are reported, averaging 0.0835 and ranging from 0.091 to 0.076 per cent., an extreme variation of 0.015 per cent.

My own results on these steels are not given, as they were not reported in time; but they add two more results by one more chemist in each case; and the results fall within the limits.

These results must be regarded as highly satisfactory, and show that here, at least, is one determination that can be made by many chemists working in many different ways, and yet with results agreeing very closely together. While it may not be necessary to determine many things as closely as phosphorus in steel, yet it would be highly satisfactory if we could do so; and this is a good standard of excellence for us to aim at.

Phosphoric Acid.—As compared with the accuracy secured in the determination of phosphorus in steel, the 1894 report of the Association of Official Agricultural Chemists* shows that on one sample 39 determinations of insoluble phosphoric acid by 18 chemists, working by the official method, showed results ranging from 0.45 to 0.03, with an average of 0.27 per cent., the extreme variation being 0.42 per cent., or over 1½ times the average determination.

By another method, on the same sample, 36 determinations by 19 chemists showed results varying from 0.34 to 0.04, with an average of 0.19 per cent.; the extreme variation being 0.30 per cent., or over 1½ times the average.

We have thus 75 determinations by 19 chemists, working by two methods, showing results ranging from 0.45 to 0.03, with an average of 0.233 per cent., the extreme variation being 0.42 per cent., or nearly twice the average determination.

On another sample 33 determinations by 17 chemists working by the official method showed results ranging from 3.85 to 2.24, with an average of 2.82 per cent., the extreme varia-

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* *Proceedings of the Eleventh Annual Convention of the Association of Official Agricultural Chemists*, August 23, 24 and 25, 1894. *Bulletin* No. 43, *U. S. Department of Agriculture, Division of Chemistry*, p. 76.

tion being 1.61 per cent., or considerably more than one-half of the average.

By another method, on the same sample, 35 determinations by 17 chemists showed results ranging from 3.49 to 2.18, with an average of 2.83 per cent., the extreme variation being 1.31 per cent., or nearly one-half the average.

Summing up again, we have 68 determinations by 18 chemists, working by two methods, showing results ranging from 3.85 to 2.18, with an average of 2.82 per cent., the extreme variation being 1.67 per cent.

The same report* shows that on one sample the results of 29 determinations of citrate-soluble phosphoric acid by 14 chemists, by the direct method of Ross, varied from 2.47 to 1.04, with an average of 1.52 per cent., the extreme variation being 1.43 per cent., or nearly equal to the average of all the determinations.

On the same sample, by the official method, the results of 23 determinations by 14 chemists ranged from 2.26 to 1.18, with an average of 1.46 per cent., the extreme variation being 1.08 per cent., or over two-thirds of the average determination.

Summing up, we have 52 determinations by 14 chemists, working by two methods, ranging from 2.47 to 1.04 and averaging 1.49 per cent., the extreme variation being 1.43 per cent., or nearly equal to the average.

On another sample, 36 determinations by 15 chemists, by the direct method of Ross, ranged from 3.29 to 1.87, with an average of 2.36 per cent.; the extreme variation being 1.42 per cent., or considerably over one-half of the average determination.

On the same sample, 24 determinations by 15 chemists ranged from 3.40 to 2.08, with an average of 2.60 per cent; the extreme variation being 1.32 per cent., or a little over one-half of the average determination.

Summing up, we have 60 determinations by 15 chemists, working by two methods, ranging from 3.40 to 2.08, and averaging 2.44 per cent; the extreme variation being 1.32 per cent., or a little over one-half of the average determination.

In the determination of the total phosphoric acid† 45 deter-

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* *Ibid.*, p. 72.

† *Ibid.*, pp. 81, 82, 83.

minations, by 18 chemists, ranged from 20.67 to 19.74, with an average of 20.09 per cent., the extreme variation being 0.93 per cent. By a volumetric method, 30 determinations by 11 chemists, ranged from 20.60 to 19.83, with an average of 20.14 per cent., the extreme variation being 0.77 per cent. By another volumetric method, 21 determinations by 10 chemists ranged from 20.45 to 19.27, with an average of 19.96 per cent.; the extreme variation being 1.18 per cent.

Combining these results, we have 96 determinations by 18 chemists, working by three methods, ranging from 20.67 to 19.27, with an average of 20.08 per cent.; the extreme variation being 1.40 per cent.

Similarly, on another sample, we have 120 determinations by 22 chemists, working by the same three methods, ranging from 18.15 to 16.25, with an average of 17.26 per cent.; the extreme variations being 1.90 per cent.

Again, on another sample, we have 96 determinations by 21 chemists, working by the same three methods, ranging from 2.85 to 2.20, with an average of 2.50 per cent.; the extreme variation being 0.65 per cent.

Copper.—At the August meeting of the Institute, in 1882, Mr. W. E. C. Eustis presented a paper entitled "Comparison of Various Methods of Copper Analysis."* For the purpose of this comparison, a very complex sample was made up, containing sulphides, oxides and metallic copper, a silicate, sulphides of iron and zinc, arsenic and nickel. The paper reports 45 determinations by 17 chemists, using some eight methods. The result showed a wide variation, ranging from 53.34 to 43.90, and averaging 47.75 per cent. After throwing out a set of six results from one concern, all of which were more than 2 per cent., and two of them nearly 5 per cent. above the nearest other result, as being manifestly too high, and two results by one chemist and one method, which were more than 2 per cent. below the nearest other result, the series still ranges from 48.72 to 46.24, with an average of 47.23 per cent. and a maximum variation of 2.48 per cent., which cannot be considered very satisfactory.

The same paper reported 17 determinations by 7 chemists on borings of pig-copper. These ranged from 91.07 to 98.17 and

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* *Trans.*, xi., 120.

averaged 94.25 per cent. After throwing out two results that were nearly 3 per cent. higher than the nearest other result, and four that were over 3 per cent. below the nearest other result, the series ranges from 94.91 to 94.38, with an average of 94.69 per cent. The extreme variation of only 0.53 per cent. must be regarded as very good work, especially when we consider the character of the material.

At the Florida meeting, in March, 1895, the results of a symposium on copper and copper-matte, initiated by Dr. A. R. Ledoux, of New York City, were presented.* Eight chemists reported the copper in the matte, some in duplicate or more, as determined by electrolysis, as ranging from 55.17 to 54.50 and averaging 54.91 per cent. The extreme variation was only 0.67 per cent., and this must be regarded as satisfactory, and very much better than the results on Mr. Eustis's complex mixture.

Six chemists reported results by the cyanide method ranging from 54.8 to 50.55 per cent., all but one of the results being below the lowest electrolytic result. These cannot be regarded as satisfactory.

A plate of copper made from melted anodes was drilled, and six chemists reported the copper in the drillings, as found by the electrolytic method, as ranging from 98.45 to 97.04, and averaging 97.67 per cent., with a maximum difference of 1.41 per cent. These results are not as good as those previously reported by Mr. Eustis.

Gold and Silver in Copper-Materials.—The symposium above referred to was undertaken primarily to test methods of assaying copper-material for gold and silver. Fourteen chemists reported the silver by scorification-assay, some entirely uncorrected, some partially corrected and some corrected for both loss in slag and cupel and presence of copper in the silver button. The averaged results ranged from 135.38 to 122.88, and averaged 128.86 ounces per ton; the extreme variation being 12.5 ounces per ton, or 9.7 per cent. of the average.

Nine chemists reported ten results by combined wet and scorification methods, a few of them corrected for slag- and cupel-absorption. The averaged results ranged from 130.68 to 123.03, and averaged 127.25 ounces per ton; the extreme variation was 7.65 ounces per ton, or 5.97 per cent. of the average determination.

* *Trans.*, xxv., 250, 1000.

One chemist reported 123.6 ounces per ton by the crucible-method.

Another reported 126.2 ounces per ton by combined wet and crucible-method, corrected for slag and cupel.

Summing up, we have 26 results by 20 chemists, working by two main methods, but both of them modified in various ways, and two methods, each by a single chemist, varying from 135.38 to 122.88, and averaging 127.94 ounces per ton; the extreme variation was 12.5 ounces per ton, or 9.77 per cent. of the average determination.

In the case of the silver-assay of the copper-borings, 9 chemists reported by the scorification-method, with and without corrections. The averaged results varied from 164.05 to 154.40, and averaged 159.36 ounces per ton; the extreme variation was 9.65 ounces per ton, or 6.06 per cent. of the average.

Fifteen chemists reported 16 results by combined wet and scorification-methods, with and without corrections. The averaged results varied from 161.40 to 148.50, and averaged 156.48 ounces per ton; the extreme variation was 12.9 ounces per ton, or 8.24 per cent. of the average. A single chemist reported 161.35 ounces per ton by combined wet and crucible-process, corrected for slag and cupel.

Summing up, we have 26 determinations by 20 chemists, working by three methods, ranging from 164.35 to 148.5, and averaging 157.67 ounces per ton; the extreme variation was 15.85 ounces per ton, or 10.05 per cent. of the average determination.

Twenty chemists, working by the four methods, reported 26 results on the gold in the matte, varying from 2.41 to 1.85, and averaging 2.245 ounces per ton; the extreme variation was 0.56 ounce per ton, or 24.94 per cent. of the average.

On the gold in the copper-borings, 20 chemists, working by two main methods, each one variously modified, and the combined wet and crucible-method by a single chemist, reported 26 results, varying from 0.501 to 0.205, and averaging 0.307 ounce per ton; the extreme variation was 0.296 ounce per ton, or 96.4 per cent. of the average determination.

Potash.—In the determination of potash, the 1894 report of the Association of Official Agricultural Chemists* gives 6 de-

* Page 22.

terminations of KCl by 6 chemists, by one method, ranging from 97.79 to 99.32, with an average of 98.56 per cent.; the extreme variation being 1.53 per cent. By another method, on the same sample, 7 determinations by 7 chemists range from 97.21 to 98.86, averaging 98.16 per cent. Combining these results, we have 13 results by 7 chemists, by two methods, ranging from 97.21 to 99.32, and averaging 98.35 per cent.; the extreme variation being 2.11 per cent.

This report contains also a table of results on soil analyses,* which I quote entire:

Table of Averages.

Soil Sample No. 2.

	Provisional Method.					Hilgard Method.				
	No. included.	Average.	Highest.	Lowest.	Difference in per cent. of Average.	No. included.	Average.	Highest.	Lowest.	Difference in per cent. of Average.
		Per ct.	Per ct.	Per ct.			Per ct.	Per ct.	Per ct.	
Insoluble matter.........	14	76.874	77.733	76.19	2	12	76.619	77.910	75.880	3.8
K_2O..........................	13	.405	.510	.27	59	11	.431	.670	.250	98
CaO..........................	13	.460	.605	.3[illegible]0	53	10	.538	.680	.390	54
MgO..........................	11	.425	.589	.360	54	9	.425	.627	.320	72
Fe_2O_3..........................	9	3.504	4.26	2.955	37	7	4.347	6.400	3.300	70
Al_2O_3..........................	9	6.613	7.500	6.240	19	7	6.285	6.828	4.460	38
P_2O_5..........................	16	.496	.600	.410	38	12	.510	.660	.430	45
Fe_2O_3, Al_2O_3 and P_2O_5..	12	10.754	11,400	10.220	11	10	11.071	12.100	10.550	14
Nitrogen....................	7	.276	.290	.262	10	...	......	......	......	....
P_2O_5, Goss method......	5	.467	.493	.425	15	...	......	......	......	....

Soil Sample No. 3.

	Provisional Method.					Hilgard Method.				
	No. included.	Average.	Highest.	Lowest.	Difference in per cent. of Average.	No. included.	Average.	Highest.	Lowest.	Difference in per cent. of Average.
		Per ct.	Per ct.	Per ct.			Per ct.	Per ct.	Per ct.	
Insoluble matter........	11	80.520	81.255	79.980	1.6	10	80.448	82.010	79.47	3.1
K_2O..........................	10	.422	.500	.305	46	9	.396	.630	.240	98
CaO..........................	9	.379	.425	.300	33	8	.411	.600	.275	79
MgO..........................	8	.381	.524	.270	67	7	.369	.490	.265	61
Fe_2O_3..........................	6	3.251	4.380	2.810	62	5	3.746	4.870	3.025	49
Al_2O_3..........................	6	6.191	7.440	5.670	29	5	5.770	6.164	5.050	19
P_2O_5..........................	11	.418	.555	.350	49	9	.429	.560	.309	59
Fe_2O_3, Al_2O_3 and P_2O_5..	8	9.927	10.350	9.440	9	7	10.091	10.590	9.550	10
Nitrogen..................	6	.190	.224	.175	26	...	......	......	......	....
P_2O_5, Goss method......	4	.369	.390	.354	10	...	......	......	......	....

* Page 41.

The Bertrand-Thiel Open-Hearth Process.

BY JOSEPH HARTSHORNE, PHILADELPHIA, PA.

(Colorado Meeting, September, 1896.)

FOR something over two years past a new development of the open-hearth process has been in operation at the works of the *Prager Eisenindustrie-Gesellschaft* at Kladno, in Bohemia. It was devised and perfected by Mr. Ernst Bertrand, general superintendent, and Mr. Otto Thiel, steel-works superintendent. The results which the inventors wished to obtain were to increase the product per furnace, to reduce the amount of refractories and additions used, to enable a poorer and more varied quality of stock to be employed, to improve the quality of the material produced and to render the control of the operations and product more certain. In all of these objects they have been successful, and have thereby considerably reduced the cost of manufacture and increased the value of the finished material.

THE PLANT.

The plant at Kladno consists of one 12-ton and one 20-ton furnace. Through an incident of construction the smaller furnace stands some distance behind and to one side of the larger furnace, and at a height of about 10 feet above it. The furnaces are in the same building as the Bessemer converters, and the metal is handled in the same casting-house by means of a locomobile ladle-crane.

The relative position of the furnaces suggested the experiments by which the process was developed, and also enabled it to be thoroughly tested. Much valuable and practical work has been done in this plant, although it is not as convenient and suitable as one especially designed for the process would be. Nevertheless, the pecuniary and technical results have warranted its continuous operation.

THE PROCESS.

The process, as at present carried out at Kladno, consists

essentially in dividing the charge between the two furnaces, tapping the metal from the upper into the lower one and removing the slag from this metal during its progress from one furnace to the other.

Both furnaces are at work on the basic system, although this is not an essential feature of the process. Pig-iron high in phosphorus and silicon is charged into the upper furnace with a small portion of the scrap, if desired, and also a certain quantity of ore and limestone. The remainder of the scrap is charged into the lower furnace, together with pig-iron and a small quantity of limestone. A little ore is also added, if necessary.

The upper furnace is first charged and the metal is melted and made hot. This takes about three hours. By the end of this time the metal is hot and fluid, the silicon is all in the slag and the carbon and phosphorus are to a considerable degree eliminated. It is then ready for tapping.

The lower furnace is charged about two hours later than the upper one. By the time the heat in the upper furnace is ready for tapping the metal in the lower one is also fluid.

The primary furnace is then tapped, the slag being carefully skimmed off the metal as it passes down the trough and prevented from entering the secondary furnace. As soon as the two metals mix together a very lively reaction ensues, which quiets down in about a quarter of an hour. The phosphorus is then below 0.03 per cent. in the bath. The heat is finished by the addition of ferro-manganese or spiegel, and is ready to be tapped, if no further improvement in quality be desired. Fifteen minutes longer in the furnace brings the phosphorus below 0.02 per cent.

While the above may be called the standard practice, and the one best fitted for a new plant and ordinary conditions, it is evident that this practice may be modified very much to suit special circumstances. For instance, the transfer from the primary to the secondary furnace may be made by means of a ladle instead of a trough. Although this method of transfer is not as convenient, and will involve more loss of time and temperature, still it will probably be more readily and cheaply installed in existing plants where the furnaces are all on the same level. This is especially the case in those plants which

have no room where primary furnaces could be erected on a higher level. The loss of time and heat in this method will not be very great, certainly not great enough to affect materially the advantages of the process. As a matter of convenience and economy, however, it will generally be better, in designing a new plant, to put the furnaces on different levels and to transfer by means of a trough, since there is no difficulty in effectually preventing the slag from passing into the lower furnace.

It is evident also that the character of the charges in the two furnaces will vary materially with the local conditions affecting the supply of raw materials. Where scrap is cheap and plentiful, more will be charged into the primary furnace and the charge in the secondary one may contain no pig-iron at all. Where scrap is scarce and dear, none will be charged into the primary furnace, and only the commercially available amount will be used in the secondary furnace. In all cases the material higher in silicon, phosphorus and sulphur will be charged into the primary furnace and the available amount of purer material will be charged into the secondary furnace together with the necessary amount of other stock.

Examples of Practice.

In order to show what has been done some typical operations are here described.

The earliest charges were made up and treated as follows:

Upper Furnace.	Tons.
Mottled and gray pig-iron,	9
Steel scrap,	2
Magnetic iron ore,	1
Limestone,	1

The pig-iron contained from 1.06 to 1.2 per cent. of phosphorus, about 1 per cent. of silicon and 0.3 per cent. of manganese.

Lower Furnace.	Tons.
The same pig iron,	2
Steel scrap,	10
Limestone,	1

The lower furnace was charged one hour after the upper

one. In two and a half hours after charging the metal in the upper furnace was perfectly melted and boiling, and the phosphorus was about 0.4 per cent. This metal was then let down into the lower furnace, the charge of which had been also thoroughly melted. The slag from the upper furnace was skimmed off as it passed down the trough. In a quarter of an hour after the two charges were united the test samples showed 0.015 per cent. of phosphorus. After the addition of ferro-manganese the steel in the ingots showed 0.065 per cent. of carbon and 0.015 of phosphorus.

The other heats, made with the same raw materials, gave similar results in all cases. It was, however, found advisable to tap the upper furnace when the bath contained about 0.6 per cent. of phosphorus. The metal flowed better in the trough, owing to its higher contents of carbon and phosphorus.

Having been thoroughly tested on this material, the process was then tried on the regular pig-iron used in the basic Bessemer practice of the works. This pig contains about 2.5 per cent. of phosphorus and about 0.2 per cent. of silicon. The charges were as follows:

Upper Furnace.	Tons.
Basic Bessemer pig-iron,	8
Limestone,	0.8
Steel scrap,	3
Magnetic iron ore,	0.5
Lower Furnace.	Tons.
Steel Scrap,	10
Limestone,	0.25

Experience has shown that less limestone and ore were needed than had at first been used, even when the mixture was decidedly more phosphoric. After less than three hours the upper furnace was tapped into the lower one.

The phosphorus in the upper bath was about as before, but sometimes was as low as 0.30 to 0.20 per cent.

The resulting steel was of the same quality as before. The first heat made from this mixture contained 0.065 per cent. of carbon and 0.031 of phosphorus. The phosphorus was rarely above this amount when the bath was quiet and ready to tap, and, of course, could be brought still lower by holding the metal in the furnace a little longer.

Some heats were made with nothing but pig-iron in the upper furnace.

This practice was not continued, as scrap would have accumulated too fast, but the results were satisfactory. In these cases the charges were:

Upper Furnace.	Tons.
Basic Bessemer pig-iron,	10
Magnetic iron-ore,	1.5
Limestone,	1

Lower Furnace.	Tons.
Steel scrap,	8
Forge pig-iron,	3
Limestone,	0.75

The time taken to make the heat was about the same, and the same quality of steel resulted.

According to the practice now followed the charges are as follows:

Upper Furnace.	Tons.
Basic Bessemer pig-iron,	7.5
Steel scrap,	5.5
Limestone,	0.15
Magnetic iron-ore,	0.20

Lower Furnace.	Tons.
Basic Bessemer pig-iron,	1
Steel scrap,	8
Limestone,	0.8
Iron-ore,	none

The lower furnace is charged about two hours after the upper one. The metal in the upper furnace contains 0.6 to 0.9 per cent. of phosphorus when tapped into the lower one, while the metal contained in the lower furnace is already highly oxidized and very low in phosphorus. After uniting the two metals it takes 15 to 20 minutes to reduce the phosphorus to 0.020 per cent.

At present from 5 to 6 heats, of 22 tons of metal charged, are made at Kladno in 24 hours. The lower furnace is empty more than half of the time. It is quite evident, therefore, that with two primary furnaces at least 10 heats could be made in the 24 hours. The present capacity is from 110 to 132 tons (charged weight) in 24 hours; and this will be doubled by the

use of another 12-ton primary furnace, which is now being erected and will soon be at work.

Comparative Results of Practice.

In order to show in more concrete form the advantages derived from the new process, the tables on this and the two following pages are taken from the Kladno reports and cost-sheets for the five months immediately before and after its adoption.

Old Process.

Consumption in kilos. per 1000 kilos. of ingots.

	Sept.	Oct.	Nov.	Dec.	Jan.	Average.
Melting-coal	371.1	336.6	378.9	336.0	330.3	350.60
Heating-coal	19.4	44.0	54.7	111.0	52.7	56.30
Steam-coal	131.1	89.0	121.7	129.1	119.1	118.00
Limestone	154.0	152.0	155.0	174.0	133.2	153.64
Magnesite	31.0	22.8	27.0	24.6	14.5	23.98
Cost of basic refractories in florins	1.1	1.3	1.6	1.4	1.6	1.40
Cost of acid refractories in florins	0.5	0.9	2.0	1.1	3.8	1.66
Output of ingots per heat in metric tons	15.4	15.6	15.1	14.3	17.8	15.64
Output of ingots per 24 hours in metric tons	37.8	37.5	36.9	36.1	42.9	38.24

These figures give a clear idea as to what was accomplished during the first five months in which the process was at work. It will be seen that the coal consumed in melting was reduced 4.63 per cent.; that the coal consumed in heating up was reduced 55.80 per cent.; and that the amount consumed for steam was reduced 19.40 per cent. The amount of limestone used per 1000 kilos. of ingots, fell from 15.36 to 7.27 per cent., which is a reduction of 52.67 per cent. The amount of magnesite used, fell from 23.98 kilos. per metric ton (53.76 pounds per ton) of ingots to 11.20 kilos. per metric ton (25.09 pounds per ton), a reduction of 53.20 per cent. The cost of basic refractories was reduced 57.10 per cent., of acid refractories 47.0 per cent., and the total cost of conversion 27.50 per cent. The output increased 72.3 per cent.; but, of course, some of this

Bertrand-Thiel Process.

Consumption in kilos. per 1000 kilos. of ingots.

	Feb.	Mar.	April.	May.	June.	Average.	Decrease. Kilos.	Decrease. Per ct.
Melting coal	309.9	429.6	329.4	328.9	274.0	334.36	16.24	4.63
Heating-coal	24.2	27.7	22.2	26.4	23.7	24.84	31.46	55.80
Steam-coal	89.4	109.5	94.5	95.9	86.0	95.06	22.94	19.40
Limestone	124.2	94.6	48.9	55.8	40.0	72.70	80.94	52.68
Magnesite	18.7	11.0	8.2	12.2	5.9	11.20	12.78	53.20
Cost of basic refractories in florins	0.9	0.5	0.5	0.6	0.5	0.60	0.80	57.10
Cost of acid refractories in florins	0.9	0.9	0.5	1.1	1.0	0.88	0.78	47.00
							Increase.	
Output per heat in metric tons	18.5	18.8	19.7	19.0	18.7	18.92	3.28	21.10
Output per 24 hours in metric tons	45.5	47.7	84.5	76.1	75.9	65.94	27.70	72.30
Allowing 20.33 tons increase for the small furnace	....	....	....	....	....	....	7.37	19.26

Old Process.

Consumption in pounds—Per 2240 pounds of ingots.

	Sept.	Oct.	Nov.	Dec.	Jan.	Average.
Melting-coal	831.26	753.93	848.74	752.64	739.87	785.3
Heating-coal	43.46	98.56	122.53	248.64	118.05	126.2
Steam-coal	293.66	199.36	272.61	289.18	266.78	264.3
Limestone	334.96	340.48	347.20	339.76	298.37	344.1
Magnesite	69.44	51.07	60.48	55.10	32.48	53.7
Cost of basic refractories in dollars	0.469	0.554	0.682	0.597	0.682	0.597
Cost of acid refractories in dollars	0.213	0.384	0.853	0.469	1.621	0.708
Output of ingots per heat in tons	15.15	15.35	14.86	14.07	17.52	15.39
Output of ingots in 24 hours in tons	37.20	36.91	36.32	35.53	42.22	37.64

Bertrand-Thiel Process.

Consumption in pounds—Per 2240 pounds of ingots.

	Feb.	March.	April.	May.	June.	Aver-age.	Decrease.	
							Pounds	Per ct.
Melting-coal	694.18	962.30	737.86	736.74	613.76	748.97	36.33	4.63
Heating-coal	54.21	62.05	49.73	59.14	53.09	55.64	70.56	55.80
Steam-coal	200.26	245.28	211.68	214.82	192.64	212.93	51.37	19.40
Limestone	278.21	211.90	109.54	124.99	89.60	162.85	181.25	52.68
Magnesite	41.89	24.64	18.37	27.33	13.22	25.09	28.61	53.20
Cost of basic refractories in dollars	0.384	0.213	0.213	0.256	0.213	0.256	0.341	57.10
Cost of acid refractories in dollars	0.384	0.384	0.213	0.469	0.426	0.375	0.333	47.00
							Increase.	
Output of ingots per heat in tons	18.21	18.50	19.39	18.70	18.40	18.64	3.25	21.10
Output of ingots per 24 hours in tons	44.78	46.95	83.16	74.90	74.70	64.89	27.25	72.30
Allowing 20 tons increase for the small furnace	....	....	....	....	....	....	7.25	19.26

was due directly to the use of a second furnace. A fair allowance for the product of this furnace would be 20 tons (20.33 metric tons) per 24 hours. If this be allowed for, the increase in product was 19.26 per cent. The increase under present practice is much greater, being about 50 per cent.

A plant of two 10-ton furnaces and one 20-ton furnace would have a capacity of at least 10 heats in 24 hours, as shown above. This would give from 200 to 240 tons of ingots. A very good product from these furnaces, run separately, would be 5 heats in forty-eight (48) hours from each, or 2½ per day. This would give 25 tons from each of the small furnaces and 50 tons from the large one, or a total of 100 tons of ingots. It seems reasonable to suppose, therefore, that the capacity of such a plant would be more than doubled, if run under the Bertrand-Thiel process.

Besides these advantages, which are shown by actual figures, it is claimed that the amounts of deoxidizers and recarburizers used are also reduced, and that the loss is smaller. The work

of the ten months contained in the table does not show any material difference between the two processes, so far as average loss is concerned; the difference in favor of the Bertrand-Thiel process being only about 0.10 per cent. It is reasonable to suppose, however, that with more experience this feature may be improved. The secondary furnace contains only a small amount of slag, which is comparatively free from oxide of iron at the end of the operation. There should be, therefore, less loss of metal as oxide; and a smaller amount of oxide in the bath, both of metal and slag, should require a smaller amount of deoxidizer to free the steel from red-shortness. This point, however, is somewhat difficult to determine, since so many specifications require more manganese in the steel than that which is necessary to make it roll well.

The control of the quality of the finished product is very complete. Steel is made to specifications within a range of from 0.06 to 1.25 per cent. of carbon. The phosphorus can be run down below 0.02 per cent. with ease; the sulphur is kept below 0.03 per cent., and the manganese can be regulated very closely. As an instance, it may be stated that a large amount of steel has been made containing 0.80 to 0.90 carbon. This steel was used for bayonets for the Italian army and the results were very satisfactory.

The work of the past year has shown that from 65 to 75 per cent. of the sulphur is removed by this process.

A Modern Silver-Lead Smelting-Plant.

BY L. S. AUSTIN, DENVER, COLO.

(Colorado Meeting, September, 1896.)

OUR successful plants in the West were at first erected on a small scale, and as their business has been built up, have been added to gradually as their limitations permitted. They have had to adapt themselves later to their circumstances, making the best of drawbacks, and putting up with much which, had it been originally allowed for, would not now impede them.

For such a plant as is now proposed, it is assumed that a site will be chosen possessing natural advantages which may be utilized to their full extent, and which, at the same time, may allow for future extension.

The arrangement of a plant on a terraced site has been generally considered the most advantageous; but many advocate a level site, claiming that it permits expansion and is convenient in many respects. Even in the case of a plant treating, say, 500 tons of material per day, and saving by terrace-arrangement 40 feet of fall, the total theoretical saving is barely one horse-power. We have on the one side, then, the level site, involving the cost of installing and operating elevators, with their attendant liability to accident,* but possessing the advantages of good ventilation, accessibility and compactness of plant; and, on the other hand, the side-hill system, with fewer elevators (they cannot be done away with altogether), increased first cost of excavation and retaining-walls, less accessibility, poor ventilation, and greater extension of plant.

The duplication of parts in a smelting-plant is, of course, very important, since delays, even for a few hours at a time, may soon occasion a greater loss than the cost of duplicating the machinery which has caused the delay. It is accordingly better, for example, to put in two cheaper and less economical engines, to insure against a shut-down, than one larger engine, which uses proportionately less fuel, but which must be occasionally stopped for adjustment or repairs.

In a plant, especially if erected upon the terrace system, the machinery-cost is not a large fraction of the total. It would, therefore, seem wise to have each machine the best of its kind, to duplicate it freely, and, by adopting the level system of building, to save the costs of extensive excavation, retaining-walls, etc., which constitute so much of the first cost of the plant, and which may occasion subsequently a heavy bill for repairs.

In operating a smelting-works the losses may be classed as follows:

* It is believed that elevators may be so thoroughly constructed as to have little liability to derangement. In any case, it would be well to duplicate this machinery, so that the half of it may be thrown out of action at any time for adjustment, without interfering with the usual operations.

1. Losses in the slag.
2. Losses in flue-dust and fume.*
3. Losses in roasting.
4. Losses by shut-downs.
5. Losses in fuel.
6. Losses in labor.
7. Losses in management.

Slag.—The losses in slag caused by incorrect feeding or charging are referred to below. That arising from improper separation of matte from slag is referred to in my remarks about separators.

Flue-Dust and Fume.—The loss in flue-dust and fume depends upon:

1. The nature of the ores.
2. The height of the smelting-column.
3. The form of the furnace.
4. The method of feeding.
5. The blast-pressure.

At present the tendency in the custom sampling-mills, and also at the smelting-works themselves, is to crush the entire lot of ore fine. Moreover, there are large amounts of fine roasted ores smelted, all of which add to the amount of flue-dust. It has been attempted to brick the finer portion of the charge, as well as the flue-dust itself, and this has been attended with a considerable degree of success. The endeavor is to make less flue-dust to begin with, since it has to be re-treated with considerable expense and loss.

That the height of the furnace influences the amount of flue-dust can be observed by the performance of the same furnace when carried low or when filled up. The high furnace work-

* In considering the losses of the materials of the charge which are borne away from the furnace by the blast or draft, it is well to distinguish between the flue-dust proper and the fume. The flue-dust may be defined as small particles of the charge borne along in the air-currents produced by the blast, and which settle out by gravity in flues, where the advance movement of the air is sufficiently slow to permit this deposition. This material resembles that of the charge, especially in its assay-value. The fume, on the contrary, arises from the metals, principally lead and zinc, which are volatilized at the temperature of reduction, and, passing from the furnace, become condensed as the result of contact with the cooler surfaces of the flues. It may thus be regarded as a cloud of smoke, floating along indefinitely, unless it comes in contact with the surface of the flue, or of the bags of the bag house.

ing upon oxidized ores can be operated almost without visible smoke, even when putting through its normal tonnage.

The form of the furnace, as respects the bosh, has an important influence, especially where the furnace is a high one, as the upward velocity of the air-currents is lessened in the ratio of the increased area of the furnace at the surface of the charge.

That the method of feeding can control the amount of flue-dust, becomes evident when we consider that by a judicious placing of the materials of the charge, we can check the velocity of the escaping gases when they become concentrated in spots, so that the blast may be diffused more evenly over the entire surface. This becomes the more important as accretions form upon the interior of the furnace.

There is a certain pressure of blast suited to a given furnace newly put in operation; and as accretions form, the suitable pressure changes. Pressures in excess of it largely increase the amount of the flue-dust without corresponding increase in the tonnage of the furnace or the cleanness of the slag.

The nature and extent of the dust-collecting arrangements will depend upon the size of the plant, the nature of the ore, and upon other considerations just named, also upon the amount of roasting to be performed. It is to be observed that, owing to the corrosive nature of the fumes arising from the roasting-furnaces, they cannot be caught in bags; and hence, to save these fumes, even in part, a system of flues would be needed. The amount of saving may in some cases hardly be sufficient to justify the increased expense, especially if it is in contemplation to put in the perfected system of bag-house and blowers. By the use of the bag-house, it is possible to dispense with extended flues and high stacks, and thus do away with one reason for the terrace-system of building.

Roasting.—Among the most serious losses in a smelting-plant are those from the roasters. According to the records of a single plant, extending over a period of five years, this loss has amounted to $40,000 per year. The temperature of the escaping gases of these furnaces was upward of 1000° Fahr., while at the stack the temperature was still 600°. At this high temperature, lead, silver and gold pass away; whereas, were these gases cooled down during their passage to the stack, we might hope to effect a notable saving. Reference will be

made to the utilization of this waste heat in connection with the discussion of the use of the hot blast.

Shut-Downs.—The losses arising from shut-downs have already been referred to. They are equal to the total expense of operation, less the fuel not used during the stoppage. All the profit of running is lost besides. Moreover, such delays are detrimental alike to the slag and to the furnaces; and the latter, when again started, do not work well until "warmed up."

It may be observed that any decided diminution of the blast, even for a short time, is a drawback, lessening the tonnage of the furnace and affecting the cleanness of the slag.

Fuel.—The losses in fuel resulting from incorrect charging may be considerable. Any way of lessening the amount of fuel to be used at the blast-furnace means not only the actual value of coke saved, but also the increased tonnage of the furnace, owing to the increase of productive charge in the same space. Excessive fuel brings in its train the disadvantages of over-fire and increased volatilization, both of lead and zinc, with the consequent more rapid formation of accretions, which eventually terminate the campaign.

Labor.—In building a plant, the endeavor should be to reduce the item of labor as much as possible; but it should not be forgotten that this must not be done at the expense of efficiency in running. Especially at night, the importance of having sufficient labor to handle the furnaces and to make repairs speedily often outweighs other considerations.

Management.—Losses in business management may arise not only from paying too much for ores, but also from buying ores containing elements injurious to the smelting-operation. They may also arise from shortages in supply, through failure to provide supplies sufficiently in advance, or in right proportion.

Concentration of Power.—The concentration of the power-generating plant under one roof and under charge of a competent engineer is a matter of much importance. In such a case the suction-fans, or blowers, for the bag-house could be so located and operated as to save the expense of an extra engineer, and perhaps of boilers. That the sampling-mill and elevators, as well as the pumps and blowers, may be thus centrally operated by wire-rope transmission or by electric motors, is undoubted.

Delivery of Materials.—Delivery of materials to the plant, including ore, fluxes, coal or coke, may be made by tramway (wire-rope or rail), by wagons, by railroad, or by two or more of these methods combined. Ore coming in quantity from a mine by tramway is conveniently dumped in a receiving-bin with an inclined bottom. Or this bin may be made in two parts, to allow of change from one lot to another while one bin is being emptied. If now it is shoveled to the crusher, an aliquot part can be reserved for a sample; otherwise the whole may be drawn off to the automatic sampler.

When the ore arrives in wagons, and especially when in small and numerous lots, the problem of handling is not so easy. Commonly the ore has been dumped from the wagons upon the ground or upon an extended floor, thence to be handled with barrows to the sampling-mill and storage-bins. This involves considerable work in moving the ore. It is undoubtedly the easiest way for the wagons to dump the ore beneath them. There is, however, considerable floor-space needed, especially where the ore is brought in wagon-trains. A more compact way is to provide bins, alongside of which the wagons unload. Even then, however, the ore must be rehandled to the sampling-mill.

Where the materials come in and the base-bullion is shipped away by rail, the trackage required will depend upon the size of the plant. It has generally been considered advantageous to have a track set at the level of the slag-dump for loading base-bullion; but as this constitutes less than one-tenth of the weight of the materials smelted, it is a question whether it would not be well to elevate the track to the receiving-level, where empties are always available, and where the tracks which transfer materials may be run direct upon the floor of the car, making use of a piece of portable curved track. This method has proved very efficient in practice.

Where, as is the case with a modern plant, all the ore is sampled as it comes in, this unloading would naturally be done at a common point, so that the ore, as unloaded, could be automatically sampled and conveniently stored. The loaded cars beyond this common point require plenty of trackage, which may be doubled or trebled, with switches to throw all cars upon the unloading-tracks before they reach the sampling-mill. The

empties are treated in the same way. The outer one of the tracks may be reserved for handling cars in or out. The track-scale is naturally set at the entrance of the yard, and should be so placed that only cars to be weighed pass over it, and not, as is so often the case, located upon the main thoroughfare. These tracks, being arranged with a continuous grade past the sampling-mill, permit the unloading and dropping past of each car without the use of switch-engine or other power.

The handling of fuel and fluxes takes a somewhat different course. Much of the coke and limestone may be unloaded direct from the cars to the furnace. The coke amounts to from one-fifth to one-eighth of the total materials, and is commonly stored at a greater distance from the blast-furnace because of its smaller relative amount, and as a precaution against fire. It has generally been stored upon the ground, but it is better to pile it upon a floor, preferably of brick, set as rowlocks, and not of wood. If care is not exercised, the coarser part of the coke may be used at one time, and the finer at another, producing corresponding irregularities in the furnace. Where the demurrage-regulations, requiring the prompt unloading of cars, are rigidly enforced, unloading-pockets may be put in to advantage. The limestone and iron-ore could certainly be thus stored, and thus much labor could be saved, especially when, as often occurs, these materials constitute one-fourth or more of the charge. The method, now quite common, of storing coal convenient to boilers and roasters is certainly practical and economical of labor, and especially when we consider that it comes in as needed, and the small amount to be carried in store cuts but little figure in the total expense of handling. The handling of coal has been treated so thoroughly in the Eastern States that we have only to look there for our best examples of practice.

Sampling and Handling.—We come now to the question of the sampling and handling of the ores.

After sampling, some of the ores, awaiting settlement, must be reserved—each lot separately. Then, when released for use, they are distributed and layered or bedded at the appropriate bins.

Where the sampling is duplicated, the mill becomes more complex. All ores are properly passed over a grizzly to the

main crusher, a precaution which has its advantage in wet or freezing weather, or when the ore is of such a nature as to be likely to clog the machinery. In some cases it is more economical to keep it away from the machinery altogether. The elevating of this crushed material to command the automatic sampler depends for its success upon a strong and thoroughly constructed endless elevator, to which plenty of height should be given.

The requirements for reliable automatic sampling are as follows:

1. The aliquot portion to be reserved for the sample should be taken from the entire stream of ore.

2. This portion should be taken out as frequently as possible.

3. It should be taken evenly from all parts of the stream.

4. The stream should be a regular one, or at least should fluctuate in a regular or gradual manner.

5. The aliquot portions taken out should be thoroughly mixed together before farther subdivision or cutting down.

6. The re-crushing of the ore should proceed as its quantity is reduced, so that the ratio of the largest rich piece to the weight of the whole shall not exceed a specified limit.

7. The machinery should be as simple and accessible as possible, that it may be easier to clean up, and that there may be consequently less danger of "salting" succeeding samples.

Commonly the sulphide-ores are crushed at once to their final size preliminary to roasting, since they require no farther treatment, and are more accurately sampled when so fine. As a general rule, all the ore of the sampling-mill should be drawn from shoots, on the principle that from the time it is dumped over the grizzly it is not to be shoveled up until it reaches the furnace. From the sampling-shoot it is to be transferred to the storage-shoot, thence to go to the roasters or to the main ore-bins, as required.

For the horizontal transfer of materials, the writer favors the use of a tram-car of good size, holding at least 2000 pounds, the loaded car traveling on a slight down-grade, the wheels (set close together to facilitate swinging on the turn-plate) having long and well-designed bearings and broad rims to the flanges. With a rate of travel of 200 feet per minute, on a straight track,

a considerable distance can be speedily covered, and curves and turn-plates should be avoided as much as possible. Accordingly, the storage-area should be covered with long and narrow bins, say 200 feet by 15 feet each, with a central track, which will permit a side-dump car to cover the entire space pretty well. As the crushed sulphide-ore is to be layered or bedded also, it is a question whether the conformation of the ground will permit it to be so stored as to deliver at hoppers on top of the roasters. One man is often selected to attend to charging the roasters; and, where they are not too numerous, has time enough to shovel up all the ore needed and to charge it to the hoppers.

Besides the material already enumerated, there remains much which has to be returned from the slag-floor, viz., part of the slag, the matte, the barrings and perhaps the flue-dust. The inclined elevator, so much used in this connection, seems out of place, since it neither takes the slag from where it is made, nor delivers it to the place where it is to be used. This can evidently be done with the vertical elevator, since a track laid to any part of the dump can connect with it, and, on the floor above, the load goes at once to its appropriate furnace. The same elevator, going to a higher level, also enables a higher track to convey the matte at once to the crushing-machinery, in preparation for roasting.

All materials of the charge may be brought to the furnaces by means of tram-cars of 2000 pounds capacity. The tracks run into the bins or alongside the railroad-cars, and are arranged to ensure convenient loading at all points. Under the method of using large charges, but little re-handling is done at the weigh-scales.

At the ore-bins, perhaps the following plan might be used: The bottom of the bin to be a movable one arranged something like the bed of a gravel-wagon. The ore drops through a movable hopper at the front edge of the movable floor into the charge-car, thus avoiding the shoveling, which constitutes so large a portion of the work.

Having arrived at the furnace with the charge, it would seem that it would be well to conform to recent practice, and to charge everything into the furnace without re-handling. But at this point we have to pause and consider. It will be recalled by some among us that in the older practice of ten or fifteen

years ago, with small and low furnaces, a good deal was thought of the skill of the feeder in nursing his furnace, feeding for a tuyere, and adding the various correctives considered at that time necessary for the welfare of the furnace. Can this, with our present large and high furnaces, be a thing of the past? It is not so considered by some of our metallurgists, who have not changed their ideas in this respect. While a furnace is in good condition, with no wall-accretions and no consequent over-fire, with a low fuel-charge to help out matters, the need of such feeding is less apparent. In the writer's opinion, however, it must be admitted that careful and proper distribution of the fuel and materials of the charge means better, smoother and more continuous running of the furnace, and a slower development of the evils we wish to guard against. Evidently, if the materials are properly proportioned and properly fed to the furnace, we may hope that the work below can be made to conform, with expectation of favorable results. Incorrect feeding and charges, on the other hand, cannot fail to show their bad results as soon as they come down.

Height of Furnace.—As regards the modern lead blast-furnace, the tendency is continually to increase its height, with the idea of improving reduction; and this (with the precautions mentioned above) is measurably true. The high shaft is certainly harder to reach, to clean off, to run down and to start up; but with long campaigns such considerations cut but little figure. It is rather with oxidized zinky ores, where high fuel seems indicated, that accretions rapidly form, and stops are accordingly frequent.

Water-Jackets.—The water-jackets of our furnaces still leave much to be wished for. If of cast-iron, they are liable to crack at any time; and even when of the more expensive steel or wrought-iron, they may burn out at some corner, and thus stop the operations of the furnace until replaced or repaired.

The old brick bosh, now entirely supplanted by the water-jacket in lead-smelting, was effective in retaining the heat, and worked smoothly when not burning out. The writer has often thought that, with boshes made up as in the recent iron blast-furnaces, the water-jackets, with their larger absorption of heat and their annoyances of cracking, could be avoided. This appears not impossible, now that the type of slag to be used

within them, being more siliceous than formerly, is less corrosive in its action on brick.

Air-Leaks.—The air-leaks about the furnace, at the various openings and joints, and especially at the tuyere-sacks, leave much to be desired. These are defects of mechanical construction, which could be remedied without difficulty, but, perhaps, at the expense of facility in handling when it is, for any reason, necessary to stop the furnace. Such stops, however, are becoming less frequent with improved machinery, and especially with the duplication of such parts of the plant as are liable to accident or need repair. With spare engines, spare boilers, and spare pumps, stoppage may be reduced to a minimum, and the small margin sometimes left between profit and loss may be saved.

Blowers.—As regards blowers, the almost universal practice has been to use the rotary blower. They have, certainly, done very good service; and the modern rotary blower has been so carefully made as greatly to reduce its "slip," though the extended surfaces of contact are still there, and leakage of the air backwards is inevitable, and especially considerable at the pressures now prevailing. The writer has long held the view that the cylinder blowing-engine is the proper one for the lead blast-furnace, since the air-joints of piston and cylinder are those of actual contact, and the metallurgist may count on his cubic feet of air, whatever his pressure. This has not, heretofore, been so evident in such work; but, with tighter joints, and by comparison with the analysis of furnace-gases and the amount of fuel used, discrepancies show themselves otherwise not to be accounted for. A whole set of purported data respecting the air used and performance of the blast, thus becomes useless; for example, deductions as to the increase or decrease of the tuyere-orifice, its pointing in a given direction, the amount of air entering the furnace, etc.

The Hot Blast in Lead-Smelting.—Since the gases at the throat of a normally-working lead blast-furnace contain about 4 to 6 per cent. of CO mingled with some 14 to 18 per cent. of CO_2 and 75 per cent. of nitrogen, and since these gases are farther diluted with air which enters at the charge-door, any endeavor to use them in combustion is out of the question. There remains, then, the alternative of heating the air-blast externally:

1. By the slag.
2. By externally heated blast-stoves.
3. By regenerative stoves, heated with oil or producer-gas.
4. By oil-residuum: either (A) in a receiver or stove, in which combustion of the oil, gas and air is effected; or (B) by oil-jets at each tuyere.

Mr. Herbert Lang has attempted the heating of the blast by means of hot slag, and claims to have obtained by this means a temperature of 500° Fahr., which, however, is much below the temperature considered necessary in iron-practice. Whether an apparatus arranged like the slag-heated boilers at Broken Hill, Australia, would prove effective in air-heating, is a question to be solved when it is conceded that the hot-blast will prove an advantage in the treatment of lead-ores.

The pistol-pipe and other hot-blast stoves, where the heat is communicated to the air by transmission through pipes, are expensive; and, because of the thickness of the pipes necessary for durability, are anything but efficient. It has been suggested by Dr. M. W. Iles that the flues leading away from the roasting-furnaces afford an important source of heat, since in this case the gases issuing from the furnace may have a temperature as high as 1000° Fahr., while at the stack they still retain 600° Fahr. Air-pipes placed within the flue would absorb a portion of the heat, at the same time cooling down the issuing gases, which, in their highly-heated condition, are carrying away value from the roasting ores. Other methods of utilizing this waste heat, such as we are familiar with in connection with puddling-furnaces, could also be applied.

The hot-blast stove of Nesmyth, of the Colorado Iron-Works, operated a year ago at the Omaha and Grant Works, at Denver, has given promise of success, with such improvement in proportions as will be suggested by farther experience. It consists of a cylindrical brick-lined receptacle, in which oil is burned in conjunction with one-fifth of the air, thus heating the remaining four-fifths needed for the blast. The products of the blast which enter the furnace are then a large proportion of air, mingled with some nitrogen and with CO_2, in case of the complete combustion of the oil. Its effect then would be to diminish the intensity of combustion, but with an increased production of CO, as the result of the reaction between the CO_2 and the incandescent coke. That

this would be objectionable in lead-smelting, one cannot say. Experience might prove it to be a great advantage. Immediately at the tuyeres, its effect ought to be to prevent the troublesome crust which so often forms upon the surface of the lead, and which is so detrimental to the working of the furnace. We would watch with much interest experiments in this line.

Dr. W. L. Austin advocates, in his own specialty of pyritic smelting, the burning of a jet of oil at each tuyere. The products are much the same as in the Nesmyth apparatus. The application is certainly very direct, simple and inexpensive; and, provided combustion can be completed before entering the furnace, it should give results as good as those of Nesmyth.

The Separation of Matte from Slag.—This is a matter which we could wish were more thoroughly settled among metallurgists. Two methods, the separating-reverberatory and the large transfer-pots, are rather suited to a plant of several furnaces than to a single furnace, where the Mathewson slag-tap, the fore-hearth, or the settling-pot are still in use. In favor of the reverberatory, it is strongly urged that the time during which the slag remains entirely liquid permits the thorough separation from it of all globules of matte and lead. By any other method, the separation has to be completed during the short time the slag remains liquid, and this, too, with the formation of a considerable share of foul slag, which has to be returned to the furnace for re-melting. The latter is not necessarily considered a drawback, as metallurgists favor the re-smelting of considerable quantities of slag, often to the extent of 40 per cent. of the charge, in our Western practice. Whether the time will come when ores will be smelted with a low fuel in the blast-furnace, and all the slag thus formed retreated in a reverberatory of the type of the Argo copper-furnace, is a curious speculation. We are certainly beginning at something which looks a little like it.

The Slag-Dump.—The slag constitutes 70 per cent. of the charge, so that the materials put into the furnace (including coke) are to the slag as 115 to 70. Hence these materials should be handled (as they are) with so much greater ease than the worthless material; and this consideration takes the precedence. It seems strange, however, that so much care should be taken to locate conveniently the place of deposit of that

worthless material (the dump), while the valuable material (the ores) is transported often a considerable distance.

A modern plant ought to provide for a cheap and easy method of slag-disposal in place of, as is sometimes the case, the slow transfer by hand to the edge of the dump. The neatest method of effecting this is, again, with the assistance of the elevator, in this case large enough to hoist a truck carrying $4\frac{1}{2}$ tons of slag, such as that used at the Omaha and Grant Works, at Denver, and now made by the Colorado Iron Works, of the same place. This truck, having been raised to the higher level, can be made to run away upon a descending grade until it reaches a bank or edge of a hill, at the right or left of the furnaces, where, turning a curve, it would come back, still descending, the track thus forming the edge of the dump, anywhere along which the contents of the pot can be discharged. The man in charge now drops the train farther back to the point of filling, ready for use once more. Any animal power is thus dispensed with; the height gained by the elevator being sufficient to insure all forward movement. Such arrangements preclude all anxieties as to dumping-room; while a small space, reserved immediately at the front of the furnace, is used in case the elevator should need repairs.

Dr. M. W. Iles, in a private communication on this subject, says:

"Other good and practical methods suggest themselves. For example, removing the slag in pots arranged trainwise, as practiced at the Arkansas Valley Smelter, at Leadville, using a compressed-air locomotive, which possesses advantages over the steam-locomotive, and which has been brought to a high degree of perfection by H. K. Porter, of Pittsburgh, Pa. Or for this might be used an electric motor, in conjunction with a trolley-system. In certain favored localities, where the slag could be removed by granulating in water, it could be conveyed to some central locality, and thence, by a simple and inexpensive bucket-hoist, placed in open cars, removed from the works and used for railroad-ballast, thus bringing an income to the company. Extreme caution should be used to thoroughly separate the matte from the slag before granulating."

Yard-Floor.—To cover the surface in front of the furnaces with cast-iron plates is a precaution not to be neglected, even though expensive. The ease of keeping clean and of moving pots about and the saving of valuable scrap-lead and matte quickly compensate for the expense; and, moreover, larger slag-pots can be used.

Roasters.—In spite of the cheaper roasting claimed for the automatic roasting-furnaces, the long-bedded roaster still holds its own, and those works using the automatic roasters retain and use the former, notwithstanding their enterprise in adopting, and skill in handling, the latter. In fact, where it is desired not only to dry-roast, but also to slag the product, it is hard to improve upon a furnace where a continually increasing heat is so skilfully supplemented by the care of the furnace-man. With ores particularly suited to dry roasting the case is different; and in this sphere the automatic roaster is unexcelled. A large class of ores, leady and containing zinc, lend themselves, however, to the operations of the long roaster as to no other. How future improvements in roasting will modify these ideas it is difficult to say. Certainly the cost of roasting has been reduced to a very low figure; and the product may be eventually so bricked or agglomerated as to remove every objection to such methods. There is here an inviting field for invention.

Roofs.—In these drier regions of the West, with infrequent rains, drainage becomes less important; still, during wet weather the disposal of water becomes sufficiently troublesome. The ores, when bedded, should be protected by suitable roofs, since sudden snows and rains so change the nature of the charge as to affect appreciably the cleanness of the slag. On the main furnace and other roofs, the bay or saw-tooth system of roofing can be used to advantage, and water can be led to the valley of the roofs, and thence by proper down-spouts to the ground or drain. The gutters can be underlaid with steam-pipes, to melt the accumulated snow or ice which forms in such places; and there will then be no question of keeping the buildings well drained, and free from danger of overflow.

Electric Mining in the Rocky Mountain Region.

BY IRVING HALE, DENVER, COLO.

(Colorado Meeting, September, 1896.)

THE superiority of electric power for mining purposes was recognized in a general way as soon as the electric motor be-

came a practical success, but it has required time and experience to amplify and fully develop its advantages and to overcome the minor difficulties that arose.

Advantages of Electric Power.

These may be considered under three heads, based on the nature of the generating power:

I. *Electricity Generated by Water-Power.*

- (*a*) Saving of coal, water for steam (an important item in many places), firemen, handling ashes, boiler-repairs.
- (*b*) Electric motors, as now made, require less attendance and repairs than steam or compressed-air engines.
- (*c*) Underground wires more convenient than pipes.
- (*d*) Avoidance of losses by steam-condensation underground.
- (*e*) Avoidance of bad effects of steam underground—heating mine, vitiating air, rotting timbers.
- (*f*) Electric motors more efficient than the small steam and compressed-air engines used on hoists, pumps, diamond-drills, etc.
- (*g*) Rotary motion of electric motor superior to reciprocating motion of engine for many purposes, especially blowers and diamond-drills.
- (*h*) Electric locomotives peculiarly adapted to underground haulage where steam is impracticable.

II. *Electricity Generated by Steam at Some Distant Point Where Fuel and Water are Cheaper.*

- (*a*) Saving of difference in cost of fuel and water between places where power is generated and used.
- (*b*), (*c*), (*d*), (*e*), (*f*), (*g*), (*h*), same as under I.
- (*i*) Superior economy of large compound and, where practicable, condensing engines, over the small, inefficient engines used on most mining machinery, proper allowance being made for losses in transforming and transmitting electric power.

III. *Electricity Generated by Steam at Place Where Power is Used.*

- (*a*) Disappears.
- (*b*), (*c*), (*d*), (*e*), (*f*), (*g*), (*h*), (*i*), same as under I. and II.

Most of the plants in this district are included in Class I.

Mr. Edward G. Stoiber's Silver Lake Mines plant, described

below, will be, when completed, an example of Class II., or rather a combination of I. and II., as a steam-plant of the highest possible economy will be used to reinforce the water-power.

The Metallic Extraction Co.'s pumping plant at Cyanide, near Florence, Colorado, used for supplying water to town and mill, is between Classes II. and III. Electricity is generated by steam in the mill and transmitted 3700 feet to the pump located at wells near the Arkansas river, avoiding, on the one hand, the carrying of steam that distance, and, on the other, the keeping of a man at the pumping-station.

Class III. is illustrated by the plants of the Pleasant Valley Coal Co. at Castle Gate and Scofield, Utah (hoisting and hauling); the Union Pacific Coal Co., Rock Springs, Wyoming (hauling); and the Colorado Fuel and Iron Co., Rouse, Colorado (pumping, ventilating and miscellaneous power), replacing a compressed-air plant.

Counter-Considerations.

Against the advantages enumerated in the preceding section must be charged interest, insurance, taxes and depreciation on the excess of cost of water-power and electric plant over a steam-plant for doing the same work; also the greater cost of attendance, if any, due to the location of machinery at two places, although this will in many cases be more than offset by the saving in attendance on motors, as compared with steam-engines and boilers.

Conditions Affecting the Cost of Plant.

The cost of an electric-transmission plant depends chiefly on three conditions:

First. Nature of water-power (assuming such power to be used) and cost of developing it.

Second. Distance of transmission.

Third. Electromotive force or voltage used.

In order to show more clearly the effect of distance and voltage on cost of plant, it may not be inappropriate to state briefly the principal electrical laws involved in the problem.

Electrical Laws and Formulæ.

(1) Electromotive force, "pressure" or voltage (symbol,

E. M. F. or E.; unit, the volt) corresponds to pressure of water in pounds per square inch, or head in feet.

(2) Current (symbol, C.; unit, the ampere) corresponds to flow of water in cubic feet per second.

(3) Power (symbol, P.; unit, the watt) corresponds to the power of falling water, and is equal to the product of electromotive force and current, just as water-power is proportional to the product of pressure or head and flow.

1 Kilowatt = 1000 watts. 1 H. P. = 746 watts. 1 Kw = $1\frac{1}{3}$ H. P. Formula, P = EC.

(4) Resistance of a conductor to the transmission of electric current (symbol, R.; unit, the ohm) corresponds to the friction of water in a pipe. This resistance is directly proportional to the length of the conductor, and inversely proportional to its area of cross-section.

(5) The electric current is equal to the electromotive force divided by the resistance. This is also analogous to the flow of water, which increases with the pressure, and decreases as the resistance or friction increases, although the law is not exactly the same.

Formula (Ohm's law), $C = \frac{E}{R}$.

(6) Energy wasted in a conductor by being converted into heat (symbol, H.; unit, the watt, as before) corresponds to the waste of power, also converted into heat, by the friction of water in a pipe, and is equal to the square of the current multiplied by the resistance.

Formula, $H = C^2R$.

Conclusions.

(A) From (4) and (6) it is evident that increasing the distance, and consequently the length of wire (other things remaining the same), will proportionately increase the resistance and the loss of power; but if the cross-section of the wire is increased in same proportion as its length, the resistance and loss will remain the same. This, however, increases the weight of the wire as the square of the distance. Hence the law: *For a given power and electromotive force (which fixes the current) the cost of copper, for a specified percentage of line-loss, varies as the square of the distance.*

(B) From (3) it appears that the same power is obtained from

high electromotive force or voltage and small current as from low voltage and proportionally large current—another analogy to water-power. But (6) shows that the loss of power varies as the square of the current, and hence *inversely* as the square of the voltage. If the loss (C^2R) is to remain the same, R can be increased as much as C^2 is decreased, or as much as E^2 is increased, which means that the cross-section and weight of the wire will be inversely proportional to E^2. Hence the law:

For a given power and distance the cost of copper, for a specified percentage of line-loss, varies inversely as the square of the voltage.

(C) Combining (A) and (B) the following law is established:

If the voltage is increased in proportion to the distance, the cost of wire for transmitting a given power with a specified line-loss remains constant.

The table on page 407 shows the cost of copper, at 14 cents per pound, per kilowatt transmitted by the 2-wire system for various distances at different voltages, with 1·0 per cent. waste of energy in line.

Considering the fact that the total cost of steam- or water-power, electric generators, switch-board and motors seldom exceeds $150 per kilowatt, it is evident that when a distance is reached that makes the cost of wire (and transformers, if used) exceed that amount, or the entire cost of the remainder of the plant, that distance may be considered to be near, if not beyond, the economical limit, unless the conditions are peculiarly favorable for electric power.

With 500 volts this condition is reached inside of three miles; with 1000 volts, inside of six miles; with 3000 volts, at about seventeen miles, and with 10,000 volts, at about fifty miles (allowing for transformers).

It is evident from the foregoing principles and figures that the key to long-distance transmission is high voltage.

Systems.

Direct Current.—Direct-current generators, suitable for power-purposes, cannot be made to operate successfully at a much higher electromotive force than 1000 volts, on account of the arcing and short-circuiting of the commutator and its connections required to rectify the current.

The direct current cannot be transformed to a higher voltage,

Volts.	Miles.										
	1.	2.	3.	4.	5.	10.	15.	20.	30.	40.	50.
500	$20.00	$80.00	$180.00	$320.00	$500.00	$2,000.00	$4,500.00	$8,000.00	$18,000.00	$32,000.00	$50,000.00
1,000	**5.00**	20.00	45.00	80.00	125.00	500.00	1,125.00	2,000.00	4,500.00	8,000.00	12,500.00
2,000	1.25	**5.00**	11.25	20.00	31.25	125.00	281.25	500.00	1,125.00	2,000.00	3,125.00
3,000	.56	2.22	**5.00**	8.89	13.89	55.56	125.00	222.22	500.00	888.89	1,388.89
4,000	.31	1.25	2.81	**5.00**	7.83	31.25	70.31	125.00	281.00	500.00	781.25
5,000	.20	.80	1.80	3.20	**5.00**	20.00	45.00	80.00	180.00	320.00	500.00
10,000	.05	.20	.45	.80	1.25	**5.00**	11.25	20.00	45.00	80.00	125.00
15,000	.02	.09	.20	.35	.56	2.22	**5.00**	8.89	20.00	35.56	55.56
20,000	.01	.05	.11	.20	.31	1.25	2.81	**5.00**	11.25	20.00	31.25
30,000	......	.02	.05	.09	.16	.56	1.25	2.22	**5.00**	8.89	13.89
40,000	......	.01	.03	.05	.08	.31	.70	1.25	2.81	**5.00**	7.81
50,000	......	......	.02	.03	.05	.20	.45	.80	1.80	3.20	**5.00**

except in a machine similar in construction to a generator and open to the same objections.

The expedient of connecting several generators in series, thus multiplying the voltage, has been tried in a few cases, but it is suitable only where power is to be transmitted and used in large units. It would be manifestly impracticable to connect several motors in series for small powers, and especially for such purposes as running hoists, pumps, blowers and other mining-machinery.

Single-Phase Alternating Current.—The single-phase alternating-current generator can be wound without difficulty for 3000 to 4000 volts, and considerably higher if necessary, as the current is taken off from two continuous rings without rectifying it, thus avoiding the difficulties experienced with the commutators of the direct-current machine.

By the principle of induction, an alternating current of moderate voltage can be transformed into a current of smaller amperage and proportionally higher voltage, for transmission, and can be re-transformed at the other end of the line to any voltage desired for lights or power, the amperage varying inversely as the voltage. The energy remains the same, excepting a small loss in the transformation, not exceeding 2 per cent. in large transformers. As the coils of the transformer are stationary, and there are no sliding-contacts, any desired amount of insulation can be used, and almost any voltage can be generated that can be controlled on the line. Many plants are in operation at 10,000 to 12,000 volts, and as high as 50,000 volts has been used experimentally with promising results.

The single-phase alternating current is widely used for lighting, but, being a simple alternating wave, is not suitable for power, as no satisfactory single-phase alternating motor of large size has yet been devised that is self-starting under load and capable of speed-regulation. If a motor built on the same lines as a single-phase generator is brought up to the proper speed by some extraneous power, so that the alternating impulses will act in the right direction at the right instants, and the current is then sent through the motor and the load gradually thrown on, it will run satisfactorily at constant speed. Such a machine is called a synchronous motor, because it runs synchronously, or in step with the alternations of the current. Its

speed cannot be regulated; and if a sudden load causes it to slow down and lose step, it stops. It is inconvenient, and in fact impracticable, for service where frequent stops and starts are necessary, because starting it is such a tedious operation, and if it must start with load on, it cannot be used at all.

Multiphase Alternating Current.—The successful development of the multiphase system during the past four years has solved the problem, and secured the advantages of both the direct and alternating currents. A multiphase generator has several windings, so placed as to generate several alternating currents differing in phase, that is, passing the zero and maximum points at different instants. Under the influence of these currents (which may be compared roughly to the cranks of a duplex or triplex engine—no dead-center), multiphase synchronous motors are self-starting under light load, while non-synchronous or induction motors will start under full load, and are capable of speed-regulation. The latter possess the good qualities of direct-current motors, and the additional advantage of having no commutator and, unless speed-regulation is required, neither collecting-rings nor brushes, the wires being simply connected to terminals on the field of the machine. On the other hand, the multiphase alternating current, like the single phase, retains the indispensable quality, for long-distance transmission, of being transformable from low to high voltage for transmission, and from high to low for use at its destination.

History.

It may be interesting to trace briefly the development of electric mining operations in the Rocky Mountain district, and the effects of the foregoing principles and systems on this evolution.

First Application of Electricity in Mines.—In July, 1888, the first electric hoist in this region, and probably in the world, was successfully started in the Veteran tunnel, Aspen, Colo. It consists of a 7½ H. P. street-car motor of one of the earliest types (just coming into use at that time), geared to a flat friction-hoist, used for hauling cars into the tunnel. Later, it was arranged so that it could be thrown into gear with either this drum or another used for hoisting from an adjacent shaft. This machine has done good work continuously for eight years, and is still in service.

Development at Aspen, Colo.—To supply this and other simi-

lar hoists that soon followed it, the Roaring Fork Electric Light and Power Company, using water-power from Hunter's creek, installed a 45-Kw. bi-polar 500-volt generator, and later a 100-Kw. generator of the same type. In 1892, this company developed another water-power on Maroon and Castle creeks, and installed two 200-Kw. multipolar 500-volt generators. This plant was started in the spring of 1893.

In 1892, the People's Light and Power Company, using water-power on Castle creek, installed a light- and power-plant, the latter consisting of four 100-Kw. bi-polar 500-volt generators.

There are now in use in the mines at Aspen thirty motors, varying in size from 1 H. P. to 120 H. P., and aggregating 622 H. P., which are used for hoisting, ventilating, diamond-drilling and running mills, samplers and miscellaneous machinery.

The successful use of electric power at Aspen was rapidly followed by many other direct-current 500-volt plants, which are enumerated in the table facing page 416.

The Virginius Plant, Ouray, Colo.—In 1890 the Caroline Mining Co., operating, at great expense for fuel, the Virginius and other mines on Mt. Sneffels, at an altitude of 12,700 feet, took advantage of the new power just coming to the front, and installed an electric plant on Canyon creek, about four miles from the mine. At this distance, wire for 500 volts would be very expensive (see table on page 407); so they boldly faced this difficulty by adopting 900 volts, a much higher pressure than had ever been used before in this kind of work. From this plant they supplied two pumps, a hoist and a blower, and ran their mills located at the mine. After the completion of the Revenue tunnel and mill, the pumps were discontinued and the mill-motors were removed to the new mill and others added. An additional plant of 300-Kw. capacity is now being installed, and an electric locomotive is to be put in the tunnel.

This plant is conspicuous in the following respects:

High altitude and precipitous nature of country.

Severity of lightning.

High voltage for direct current.

Great saving by electricity, amounting to the cost of the plant every year or two, and permitting the profitable working of the property at times when it would not otherwise have paid expenses.

The Anaconda, Montana, Transmission-Plant.—Several years ago, the Anaconda Copper Mining Co. installed a plant for transmitting power 2½ miles, to run electrolytic generators in its refinery. Eight 100-Kw. 500-volt bipolar generators were connected in two series of four each, giving 2000 volts, and at the receiving-end, eight 60-Kw. 500-volt bipolar motors were similarly connected and belted to a shaft, from which the electrolytic generators were driven. This arrangement was abandoned some time ago, because the total capacity of the water-power plant was required for light and power in the town of Anaconda; and a high-economy steam-plant was therefore installed at the refinery.

The Telluride Plant.—In 1891 the San Miguel Consolidated Mining Co., of Telluride, Colo., developed a water-power at the junction of the Lake and Howard forks of the San Miguel river, for the purpose of supplying power and light to mines and mills in that district, at distances varying from two to fifteen miles. The direct current at 500 or even 1000 volts being too expensive for that distance, they turned to the alternating current; and the only alternating current available at that time being the single-phase, this was adopted. Several mills were run by synchronous motors quite successfully; but the application of power to miscellaneous purposes, requiring speed-regulation and frequent starts and stops, was impracticable with this system, for the reasons heretofore explained. The plant has recently been changed to the multiphase alternating system, using two-phase generators and motors, and an arrangement of transformers to change the two-phase into three-phase current for transmission, on account of the saving of 25 per cent. in copper effected by the three-phase system, as compared with the two-phase. This plant is now lighting the town of Telluride and supplying motors for running a number of mills in that vicinity.

Silver Lake Mines Plant, Silverton, Colorado.—In 1894 Mr. Edward G. Stoiber, owner of the Silver Lake mines, installed the first multiphase plant in this part of the country. The three-phase system was adopted in preference to the two-phase, partly on account of saving of 25 per cent. of copper in line, and partly because of the much larger number of plants on the former system in successful practical operation. The capacity

of the plant was doubled in 1895 and is now being still farther increased. This is the largest and most complete three-phase plant for purely mining purposes in the United States, and operates almost every variety of mining-machinery, including concentrating-mill, hoist, air-compressor for drills, pumps, blowers, machine-shop and lights. The distance of transmission is 3 miles. The available water-power being insufficient for the needs of the mine, especially in winter, and the generating-station being on the railroad, where coal can be delivered at a moderate price, it is intended to install a compound condensing steam-plant of considerably larger capacity than the present water-power plant, the engines and water-wheels being arranged to work together so that the water-power can at all times be utilized up to its limit, the remainder of the required power being furnished by steam.

Of the large number of electric-mining plants installed in the Rocky Mountain district during the past eight years, the foregoing are selected in tracing the development of electric power, as they illustrate all of the systems that have been tried, namely:

Direct current, moderate voltage (500 volts or less).
Direct current, high voltage (900).
Direct current, using several generators and motors in series.
Single-phase alternating.
Two-phase alternating.
Three-phase alternating.

The table facing page 416 gives a list, which is believed to be complete and correct, of all electric-power plants for mining and ore-reducing purposes in this district, showing in detail the number, kind and capacity of the machines installed and the distance of transmission.

Electric Mining Machinery.

As will be seen from an examination of the appended list of plants, electricity is being applied to the operation of every kind of machinery used in mines.

Hoists.—The first application of electric power, and one of the simplest, was to hoisting; the rotary motion of the electric motor being easily adapted to this work.

Most of the earlier machines consisted of street-car motors,

geared to flat-friction or V-friction hoists. This type is very satisfactory for small or medium-sized machines, as the friction-gear is an assistance to the motor-controller in smooth starting.

For large hoists a positive geared motor is more reliable; but it is desirable to interpose a friction-clutch or equivalent device at some point between armature and drum, as a safeguard in case of excessive strain on gearing, caused by the inertia of the armature when the drum is stopped by a too sudden application of the brakes. Mr. D. W. Brunton, of Aspen, has designed a slipping-pinion, which is used on the electric hoists in mines under his management, and serves this purpose admirably.

The choice of the best kind of motor depends considerably on the size of the hoist, its location, and the nature of the work. For an unbalanced hoist of moderate size, especially if placed underground and exposed to dirt and water, the iron-clad series-wound street-car type is well adapted, as it is strong, well protected and designed to stand heavy work on intermittent service. In this motor, efficiency, low heating and absolute freedom from sparking are to some extent sacrificed for compactness and lightness. For large hoists, which are generally located in comparatively clean, dry places, and, if over-balanced, work almost continuously, hoisting and lowering, and in which high efficiency is more important than in small hoists, the stationary type of motor is usually preferable.

The speed-controller is one of the most important features of an electric hoist. On many of the earlier hoists the commutated field, thrown into various combinations of different resistances by a cylinder-switch, was employed; this form of control being at that time widely used in street-car service. This controller gave quite satisfactory results when assisted by friction-gearing; but with positive gearing it would not give a sufficiently gradual start. On most hoists a variable resistance in armature-circuit is employed; and by making this resistance sufficiently high, a perfectly smooth start may be obtained, even with slack rope. The most satisfactory rheostatic controller, especially for heavy work, is one in which the resistance is cut in and out by a cylindrical switch with magnetic blow-out, which avoids the troublesome effect of arcing at contacts, when the current is broken.

In some cases it is practicable to use a double-motor equipment, with series-parallel controller, such as is now employed almost exclusively in street-car work.

By overbalancing a hoist, making the counterweight equal to the dead load plus about half the live load, the work in hoisting and lowering can be made approximately equal, and the maximum current and size of motor can be reduced to considerably less than half of what would be required for doing the same work with an unbalanced hoist. This principle is used in the electric hoists at the Free Silver, Alta Argent and Della S. mines at Aspen, and the Silver Lake mines at Silverton, and will doubtless be employed more generally in the future than in the past.

The Alta Argent hoist, in addition to being overbalanced, is arranged in a novel manner. The hoist, with its motor, is placed on a substantial platform above the head of the incline, and is out of the way of the operator, who is located, with the controlling levers, just below the hoist and on the level where the cars are run off. This gives more room for handling the cars as they are raised, and enables the hoist-operator to do this work.

The largest electric hoist in the world is on the Free Silver shaft at Aspen, Colorado. It is an overbalanced double-reel, flat-rope hoist, equipped with a 100-Kw. 4-pole motor, capable of developing 120 H. P. continuously and 150 to 175 H. P. intermittently. An auxiliary 60-H. P. motor, ordinarily doing other work, is arranged so that it can be geared to the hoist counter-shaft and assist the large motor if necessary. Each reel has 1500 feet of 4-by-$\frac{3}{8}$-inch flat rope, weighing $2\frac{3}{4}$ pounds per foot. The cage weighs 1375 pounds, bucket and ore (hung from cage in sinking) about 2400 pounds, car and ore about 3500 pounds, and bailer, containing 111 cubic feet of water, about 9000 pounds. Two counter-weights are used, the one for cage and bucket weighing 2450 pounds, and the one for cage and car 2675 pounds, the two being combined when bailing. Two armature-pinions are provided (the motor sliding on rails), one for ordinary hoisting giving a speed of 600 feet per minute, and the other for bailing at 1000 feet per minute, which, with the maximum load of about 10,000 pounds (bailing), would require over 300 H. P. net, exclusive of friction in

gearing and rope, were it not for the overbalanced arrangement. Actual tests have shown that the average current consumed in hoisting with counter-weight is only about one-third of the amount required when the hoist is unbalanced.

Pumps.—The electric pump presents a somewhat more difficult mechanical problem than the hoist, on account of the conversion of rotary into reciprocating motion, and the importance, in most cases, of compactness and protection of the motor against water. That the problem has been satisfactorily solved, however, is proved by the large number of electric pumps of various kinds and sizes in successful operation.

Duplex and triplex pumps, both vertical and horizontal, are suitable for operation by electric motors. For small pumps, where plenty of space is available, a belted motor affords the cheapest arrangement and gives satisfactory results. In the majority of cases a geared pump with motor on the same base is best. Both spur- and worm-gears have been used successfully, various devices being employed with the latter to neutralize the thrust. In the 70-H. P. pump in the Virginius mine (Caroline Mining Co., Ouray), the armature-shaft carries two worms, one right- and one left-hand, working into two gears which mesh into each other. In the 15-H. P. pump in the same mine, the armature is placed vertically, and its weight approximately offsets the thrust. In most of the electric pumps made during the past few years, spur-gearing has been used, and with good results.

The sinking-pump is the most difficult to design, on account of the small space available for the motor, and the necessity of enclosing it in a practically water-tight case; but numerous successful pumps of this kind have been made and installed. The three-phase induction-motor is specially adapted to this work, as it has no brushes or moving contacts, and the wires can be carried through water-tight bushings in the case to the stationary terminals on the field.

Speed-control is an important question in electric pumping. Where waste of power is unobjectionable, a rheostat in armature-circuit is suitable. If high efficiency is imperative, and the required variation in speed is not great, it may be economically accomplished by varying the field-strength of the motor, either by commutating a sectional field or by the use of a rheostat. When the generator supplies nothing but the pump, an excel-

lent method is to vary the generator-voltage by changing its speed or field-strength, or both. In some cases a water by-pass can be advantageously used. In others it is best to pump at full capacity intermittently. The most suitable method is a matter of judgment in each case.

Blowers—The running of blowers and exhausters is another simple operation, the motor being either belted, geared or direct-connected to the blower-shaft. Small outfits of this kind, placed at various points throughout the mine, run continuously with very little attention, and afford the most economical and satisfactory ventilation—far superior to the vitiated air that has passed through air-compressors and drills.

Percussion-Drills.—To obtain with electricity the rapid reciprocating motion with varying stroke and necessary elasticity required in a percussion-drill, and at the same time get a machine that will stand unlimited abuse, has been the hardest problem in the mining field that the electrician has had to solve.

Two general methods have been followed. One employs the ordinary rotary motor, connected to the drill by a flexible shaft and producing the oscillatory motion by cranks, cams, levers, springs and similar devices. The other uses the solenoid principle, the plunger being moved back and forth within two solenoids, placed end to end, by currents sent through the two alternately, these currents being shifted automatically at the drill or generator—generally and preferably at the latter.

The solenoid type of drill is the only one that has been used commercially and successfully in this country. These drills were tried in several mines in Colorado and elsewhere about four years ago, but were only partially satisfactory. The principal defects were lack of pulling-power, heating of solenoids, unsoldering of connections and breaking of drill-chucks, due to the crystallization of the bronze of which they were at that time made. These defects have been remedied by better electrical and mechanical design of solenoids and connections, and the adoption of an all-steel plunger and chuck. The improved drills have been used successfully for some time in quarrying and tunneling in the East; and two plants, as shown in the table, have recently been installed in mines in this vicinity, the operation of which will throw much light on the exact value of the drill for mining use.

Diamond Prospecting-Drills.—In contrast with the percussion-

drill, the rotary diamond-drill is more easily and satisfactorily operated by an electric motor than by any other power, not only because the rotary motion of the motor can be applied directly to the drill, while the reciprocating motion of a steam- or air- engine must be converted, but also because the former is much steadier than the latter, and causes less wear and breakage of carbons. Moreover, the transmission of power into a mine by electricity is more economical than by steam or air, and the electric drill, with its wires, can be more conveniently moved from place to place than the steam- or air-drill and pipes. On account of these advantages the electric diamond-drill has become a favorite wherever current can be conveniently obtained.

Coal-Cutters.—Electricity was successfully used for coal-cutting even when the direct-current motor was the only kind available. The application to the chain-cutter of the three-phase induction-motor, without commutator, collector or brushes, and avoiding all possibility of sparking, has produced an almost ideal machine for this work. Another valuable characteristic of the induction-motor for this kind of service is the fact that, when overloaded beyond a certain limit by striking a hard stratum or by sticking of the cutter, it stops, thus relieving both motor and machine from excessive strains.

These coal-cutters are being rapidly introduced in the East, but have not yet been used in this part of the country.

Locomotives.—Electric haulage in mines, under ordinary conditions and where distance and tonnage are not too small, is without much question superior to any other system available. It is more flexible than rope-haulage and more economical than mules or compressed air.

Electric locomotives have been principally employed in coal-mines on account of larger tonnage and longer hauls; but, as shown in the appended list of plants, are coming into use in metalliferous mines, and will be used more widely as long tunnels, tapping numerous veins, become more common.

Mine-haulage is similar in most respects to street-railway service; but the locomotives must generally be adapted to narrower gauge, the motors should be better protected, the speed is slower and the weight on driving-wheels greater.

The series-parallel controller, now so widely used in street-

car service, which throws the motors in series for starting and slow speeds, and in parallel for higher speeds, is not so suitable for a mining-locomotive, for the reason that the slipping of one pair of wheels (which is more liable to occur than on a street-car, the load being behind the locomotive and the rails being more slippery) increases the speed of the motor on that axle, raises the counter-electromotive force and cuts down the current through both motors, if in series, thus reducing the power of both. For this reason some form of rheostat-controller is generally used.

The essential requirements in a mining-locomotive are compactness, strength, simplicity, convenience of handling and especially durability and freedom from repairs.

Placer-Mining Machines.—Electric power has been very satisfactorily applied to the operation of placer-mining machines, several motors being employed to run the different parts of the dredge and gravel-treating apparatus. One motor generally raises and lowers the dipper; one forces it into the bank; another works the turn-table for depositing the gravel in the hopper; and a fourth operates the cylinder, tailings-carrier, amalgamator, etc. All of these motors are supplied by a generator located at any convenient point. The peculiar adaptability of electric motors to such work is evident; in fact it would be difficult to operate such a machine in any other way.

The Bennett Amalgamator Manufacturing Co. is operating a plant of this kind for the South Park Mining Co. at Green River, Utah, and the Gold Dredging Co. has a similar installation at Bannock, Montana.

Electro-Metallurgy.

Copper-Refining.—The principal application of electricity to metallurgical operations in this part of the country is copper-refining, which is carried on extensively at Anaconda and Great Falls, Montana. The plant at Anaconda consists of eight electrolytic generators, aggregating 1870 Kw., and its capacity is 125 tons of refined copper per day, the average output being about 100 tons per day. They also have a 100-Kw., 110-volt power-generator, supplying two 25 H.-P. locomotives and three traveling cranes, for handling and transporting materials and product.

The commercial success of the process is too well established to require discussion. Its economy is principally a question of the cost of power and the scale on which operations can be conducted.

Gold- and Silver-Extraction.—Electricity has been applied as an auxiliary in various processes for the treatment of gold and silver ores by cyanide, chlorination and amalgamation. While some of these processes are based solely on the faith of the inventor that a current of electricity sent through his mixture will in some mysterious way produce results that he cannot otherwise obtain, others undoubtedly possess merit, as they are founded on well-established laws of electro-chemical action. These processes are usually conducted with considerable secrecy, and it is impossible to give any reliable data as to the actual results accomplished or the economy of the operations. There is reason to believe, however, that, with the proper combination of chemical, metallurgical and electrical knowledge brought to bear upon this subject, good results may be expected.

Future Development.

During the past eight years, and principally in the last four years, 52 distinct companies in the Rocky Mountain district alone have installed electric-power machinery for mining and ore-reducing purposes, comprising 71 generators, aggregating 10,964 kilowatts, and 135 motors, aggregating 5201 horse-power, operating every variety of mining and milling machinery.

Electric apparatus, formerly regarded as delicate and peculiarly subject to break-downs, has been brought to such a degree of perfection that depreciation and repairs may be considered as less on this than on almost any other kind of machinery.

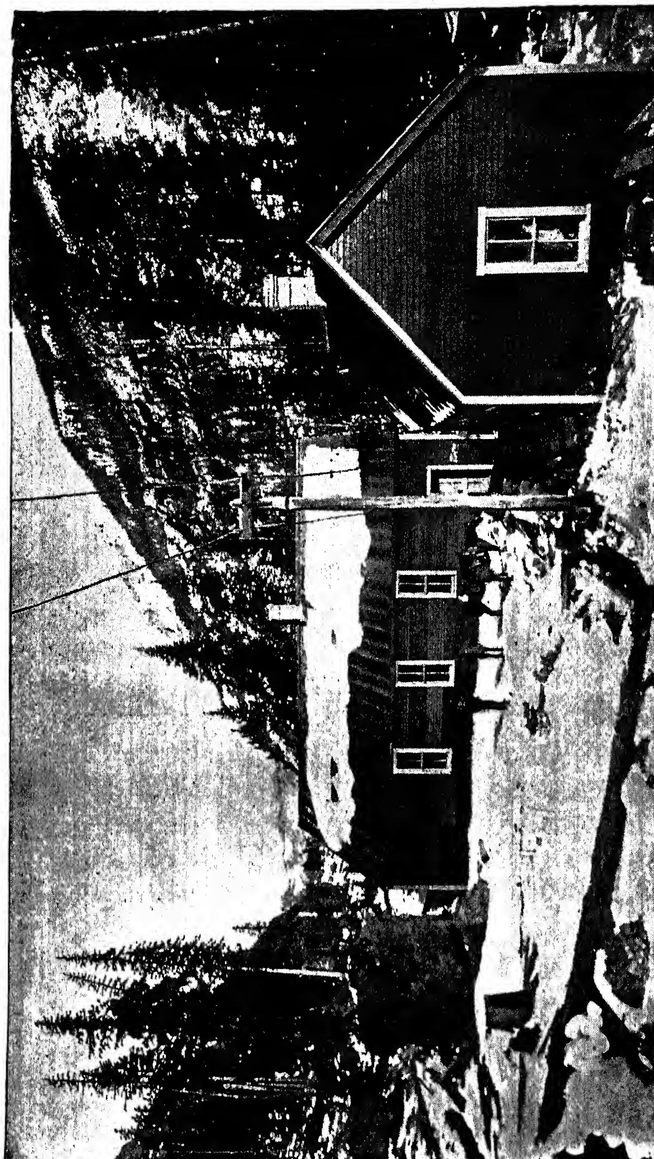
The multiphase high-voltage system has brought nearly every mining district within economical reach of water-power.

The induction-motor, without commutator, collector or brushes, is the acme of simplicity and durability.

Electro-metallurgical operations are increasing and give promise of success.

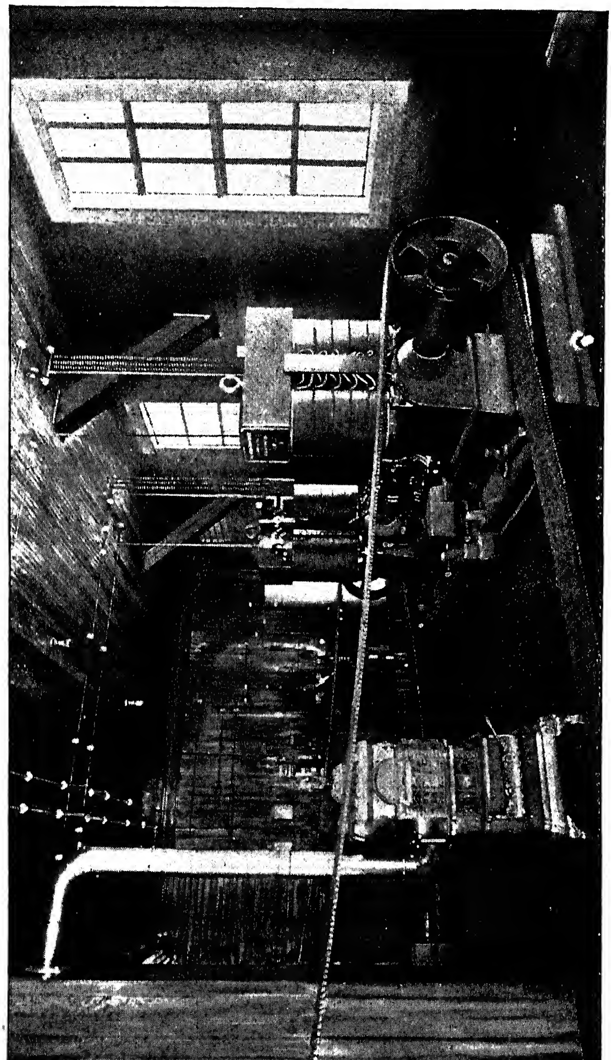
Under these conditions it may be safely predicted that during the next five years much greater progress will be made, and the application of electricity will become one of the most attractive and important features of mining economy.

Fig. 1.



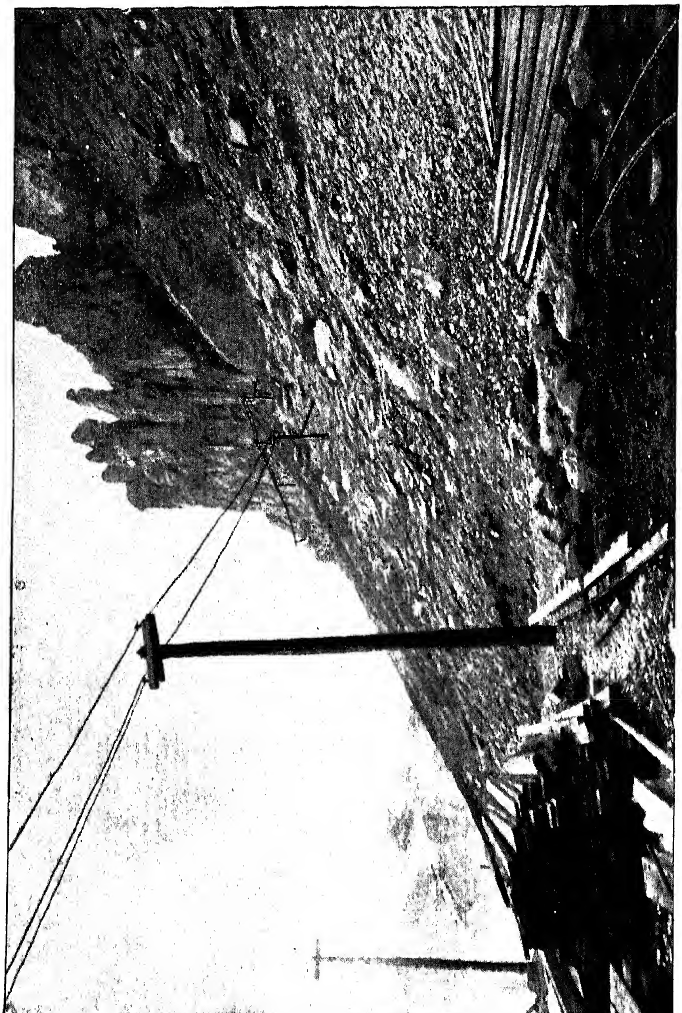
POWER-HOUSE,
Caroline Mining Company's Plant,
(Virginius Mine and Revenue Tunnel),
Ouray, Colorado.

FIG. 2.



INTERIOR OF POWER-HOUSE,
Caroline Mining Company's Plant,
Ouray, Colorado.

Fig. 3.



UPPER END OF POLE-LINE,
Viewed from Virginius Mine (altitude 12,700 feet),
Caroline Mining Company's Plant,
Ouray, Colorado.

FIG. 4.

POWER-HOUSE,
Silver Lake Mines Plant,
Silverton, Colorado.

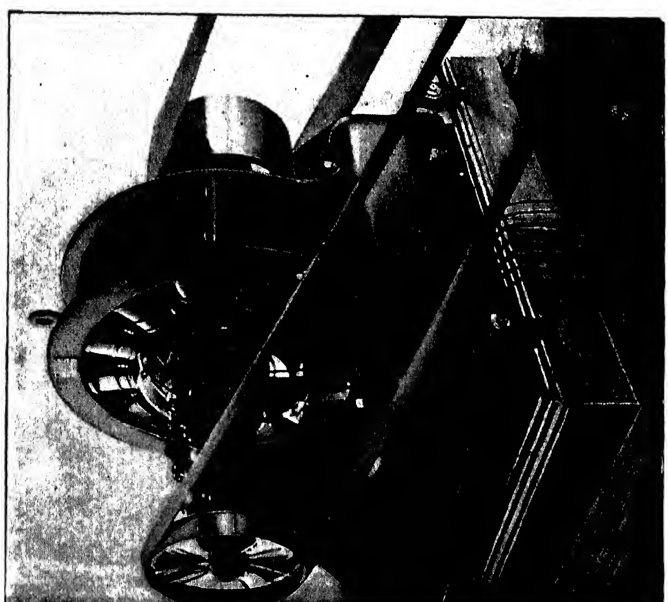
FIG. 5.

TRAIL TO ONE OF SILVER LAKE MINES.

Fig. 6.

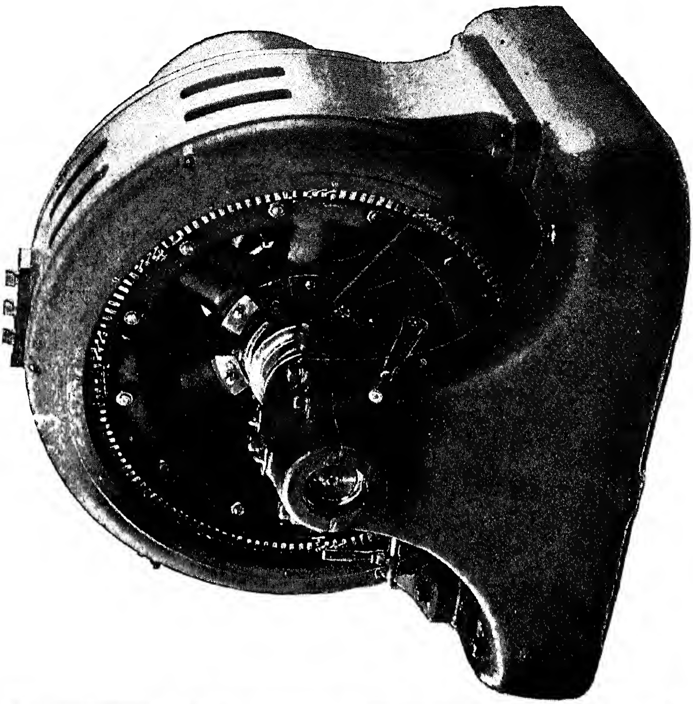
SILVER LAKE MINE AND MILL, SILVERTON, COLORADO.
Altitude 12,300 feet.
Operated by Electric Power, Three-phase System.

Fig. 7.



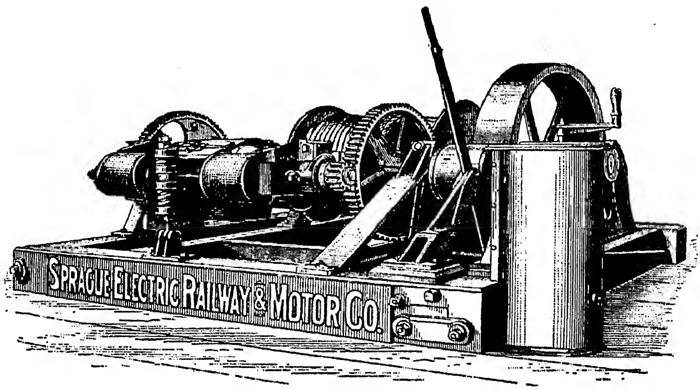
ONE OF THE 150 K. W. THREE-PHASE GENERATORS.
Silver Lake Mines Plant,
Silverton, Colorado.

Fig. 8.



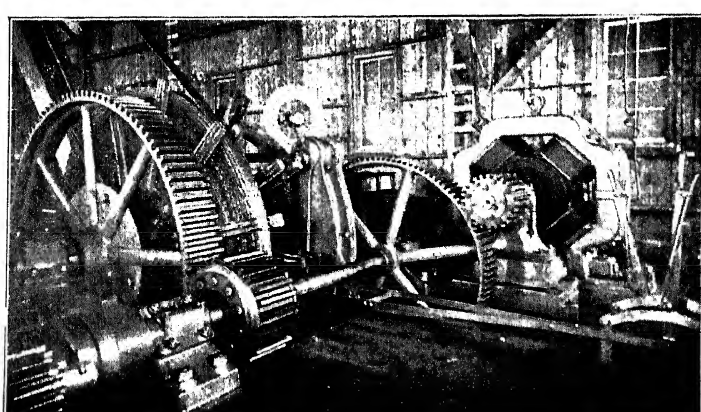
ONE OF THE 100 H. P. THREE-PHASE INDUCTION MOTORS.
Operating Silver Lake Mill and Air-Compressor in Mine.

FIG. 9.



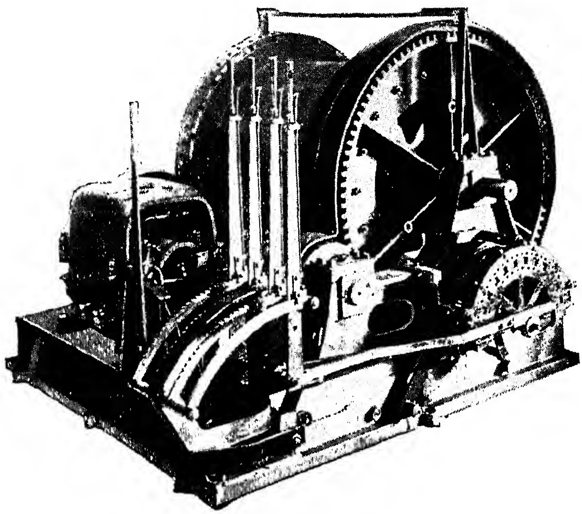
FIRST ELECTRIC HOIST IN AMERICA.
Equipped with 7½ H. P. Motor.
Capacity 1500 pounds, 100 feet per minute.
Installed in Veteran Tunnel, Aspen, Colo., July, 1888.

FIG. 10.



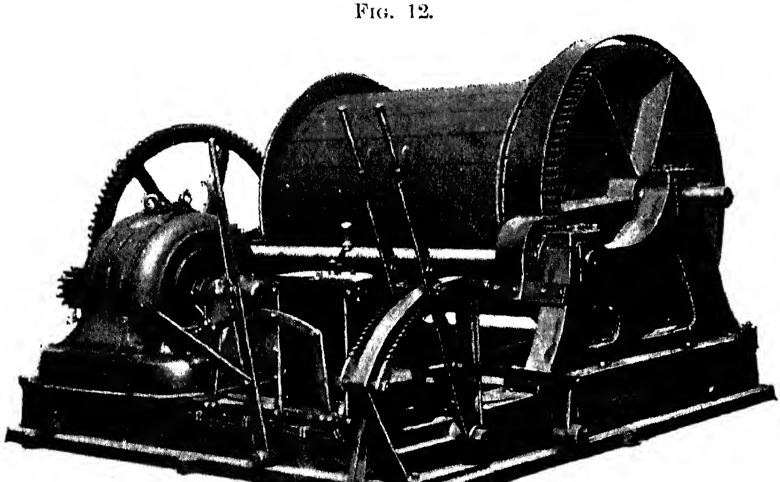
LARGEST ELECTRIC HOIST IN AMERICA.
Equipped with 150 H. P. Motor,
and Auxiliary 60 H. P. Motor.
Capacity, over-balanced, 10,000 pounds, 1000 feet per minute.
Installed at Free Silver Shaft, Aspen, Colo., Dec., 1894.

FIG. 11.



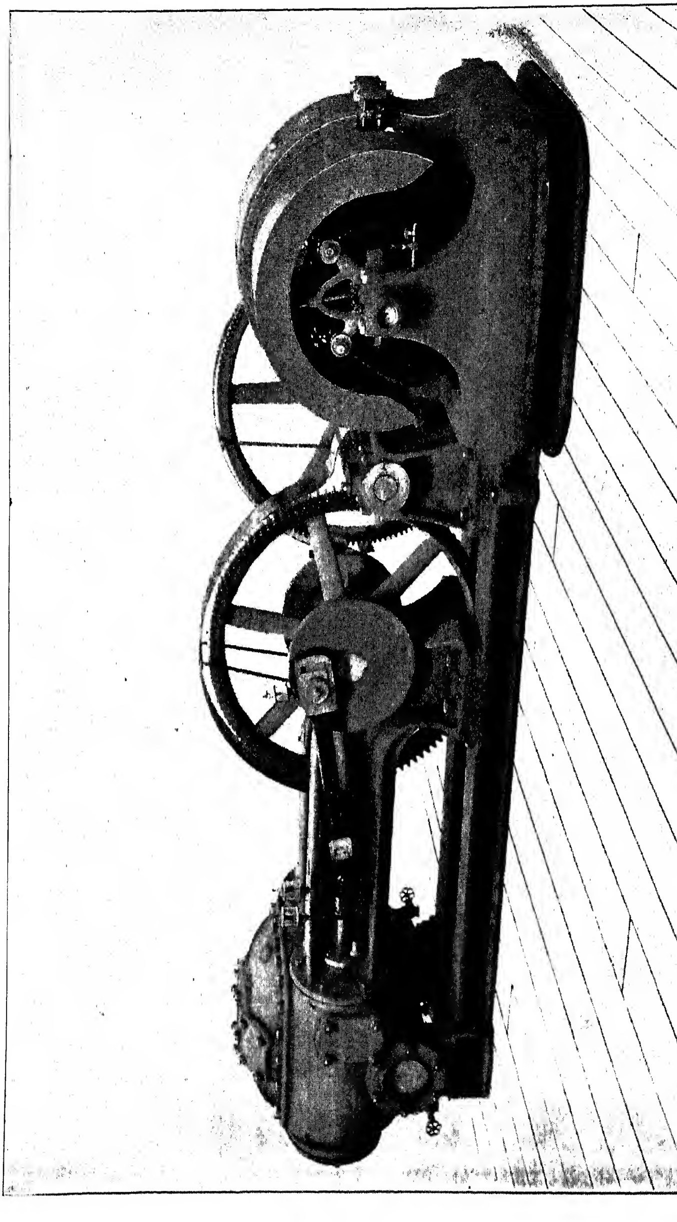
ELECTRIC HOIST,
Equipped with
Direct Current Railway Type Motor,
Pleasant Valley Coal Co., Castle Gate, Utah.

FIG. 12.



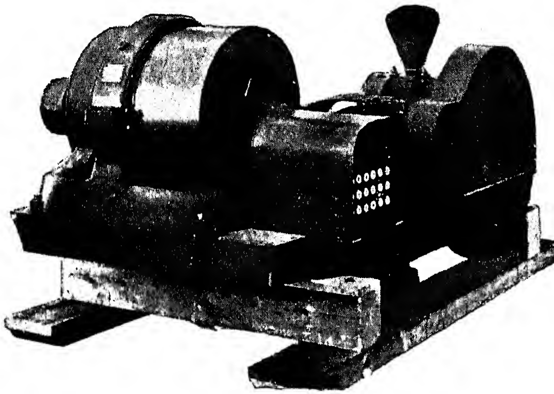
ELECTRIC HOIST,
Equipped with
Three-phase Induction Motor.

Fig. 13.



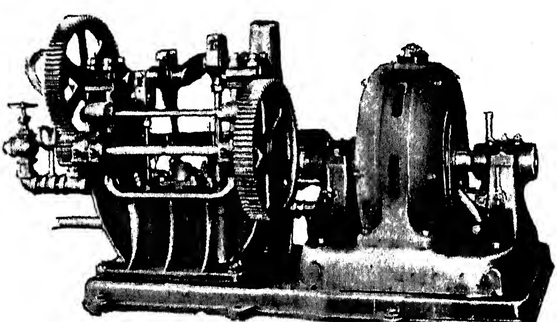
HORIZONTAL DUPLEX 7" x 10" PUMP.
Equipped with
15 H. P. Slow-Speed Motor.
Colorado Fuel and Iron Company's Plant, Rouse, Colorado.

FIG. 14.



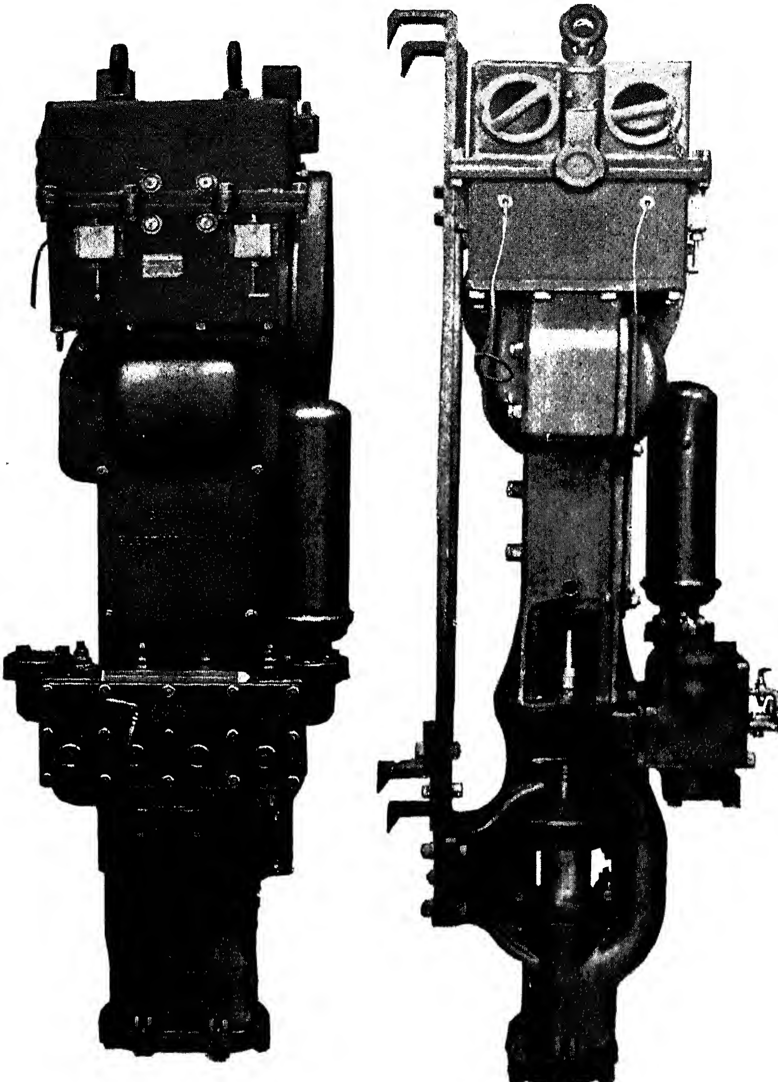
TRIPLEX TRACK-PUMP.
Equipped with
Direct-Current Slow-Speed Motor.

FIG. 15.



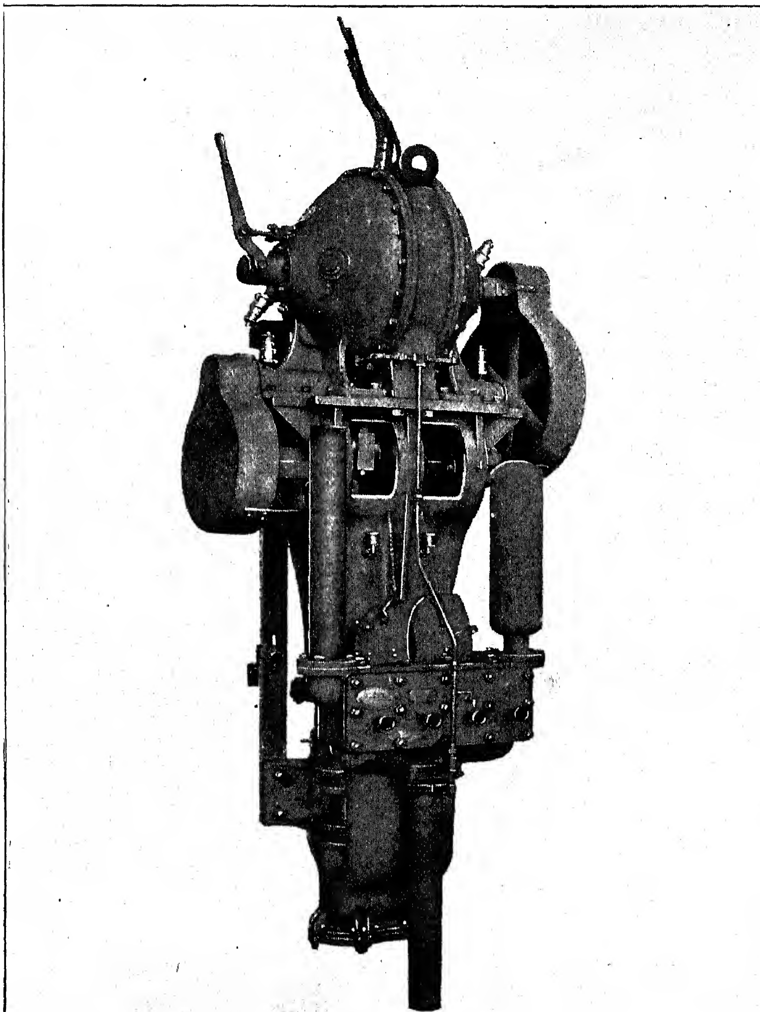
VERTICAL TRIPLEX PUMP.
Equipped with
Three-phase Induction Motor.

Fig. 16.



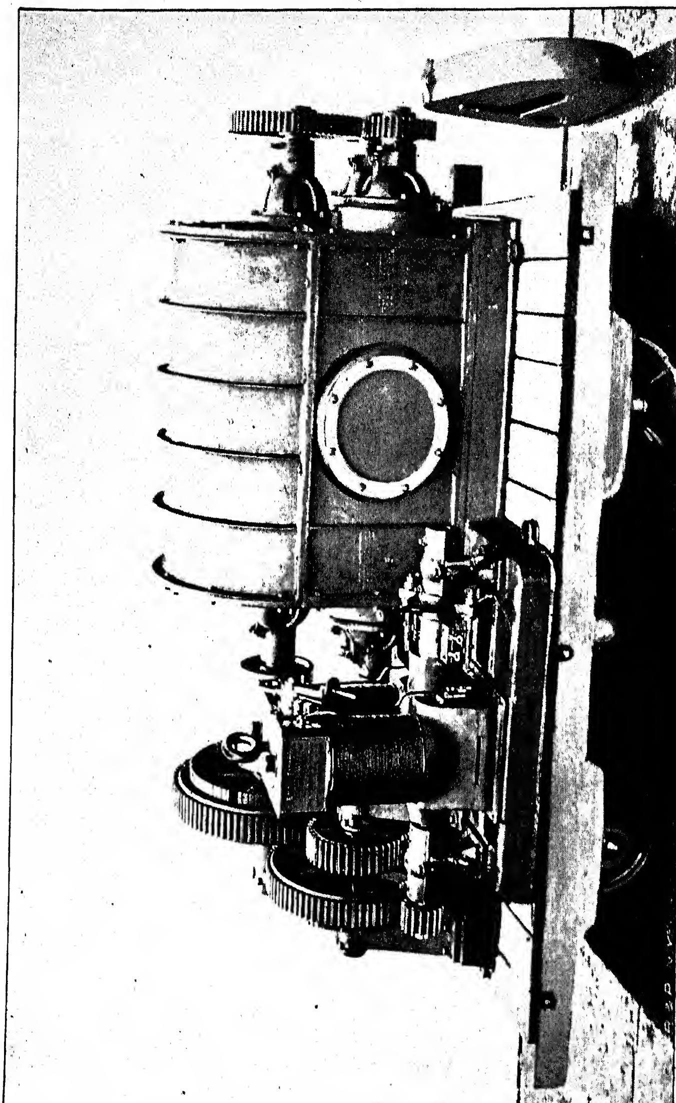
DUPLEX SINKING-PUMP.
Equipped with
Direct-Current Motor in Water-tight Casing.

FIG. 17.



DUPLEX SINKING-PUMP.
Equipped with
Three-phase Induction Motor in Water-tight Casing.

Fig. 18.



BLOWER AND DIRECT CURRENT MOTOR.
Type used in
Virginius and American-Nettie Mines, Ouray, Colorado.

Fig. 19.

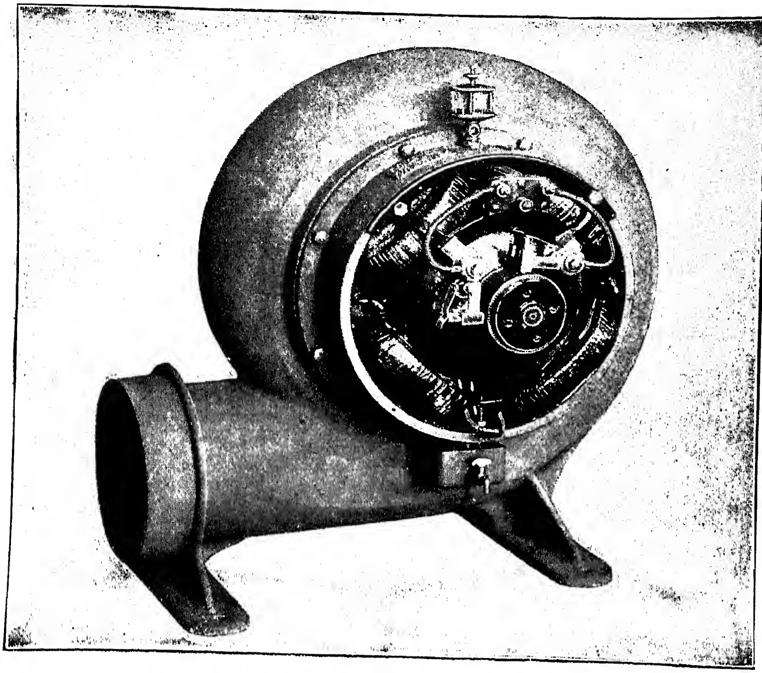
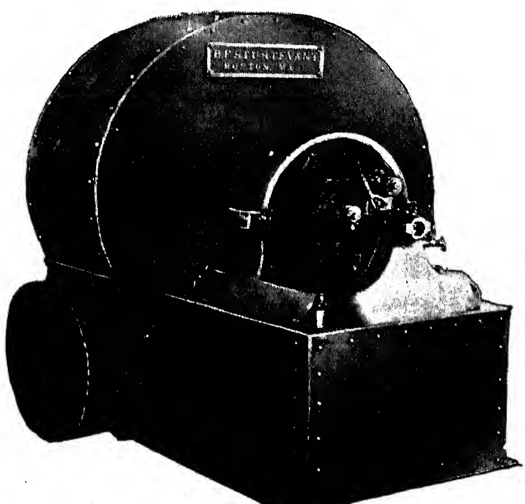
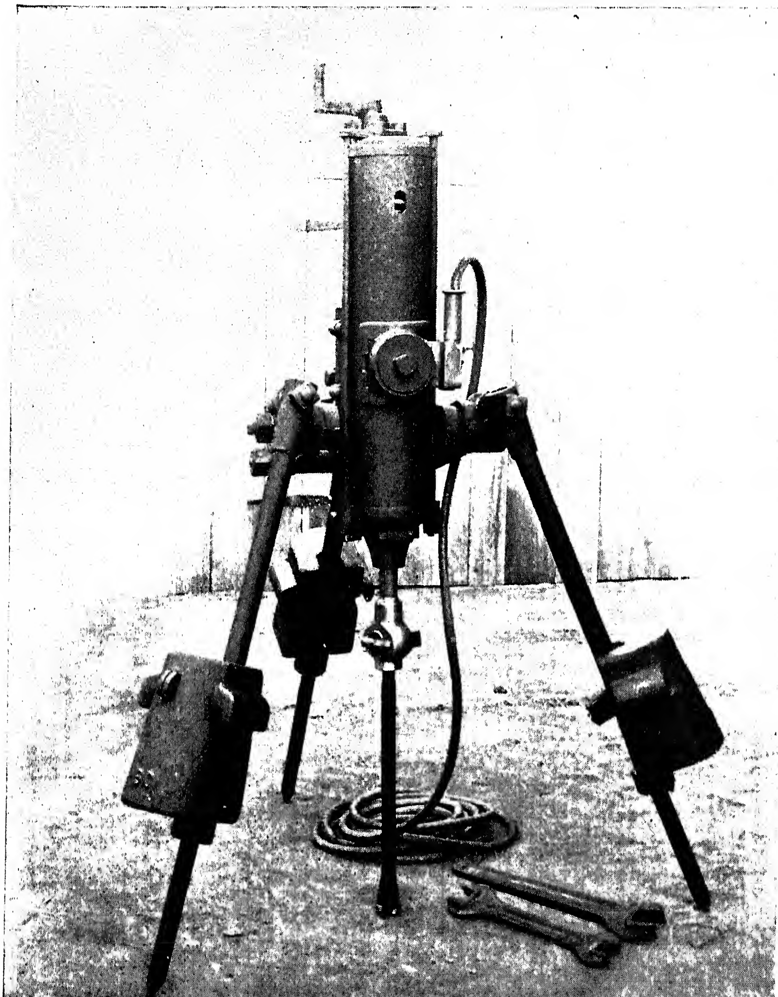


Fig. 20.



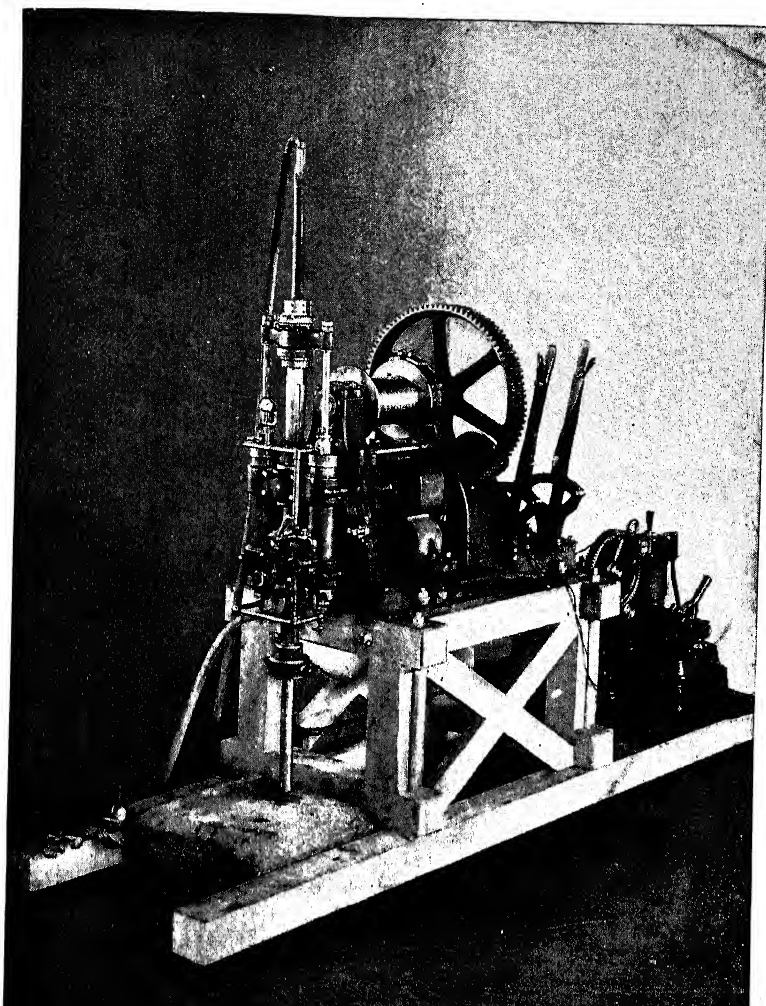
TWO TYPES OF BLOWERS.
Equipped with
Direct-Connected Slow-Speed Motors.

FIG. 21.



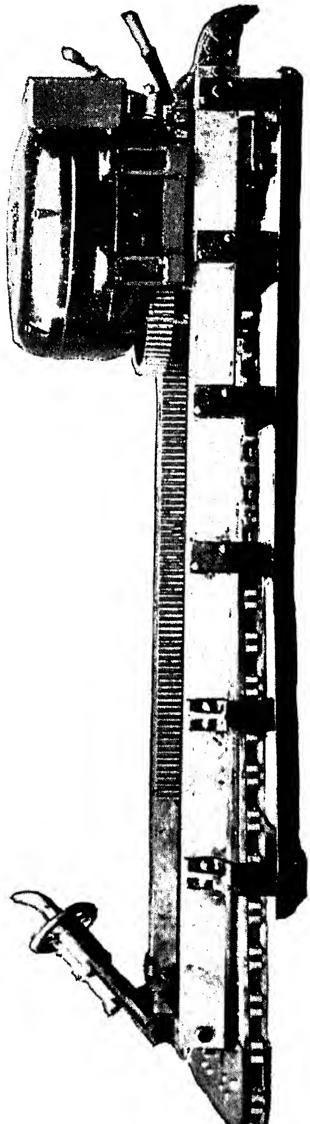
ELECTRIC SOLENOID PERCUSSION-DRILL.

FIG. 22.



ELECTRIC DIAMOND PROSPECTING DRILL.
With Electric Pump for Flushing Hole and Operating Hydraulic Feed, San Bernardo Mine, Trout Lake, Colo.

FIG. 23.

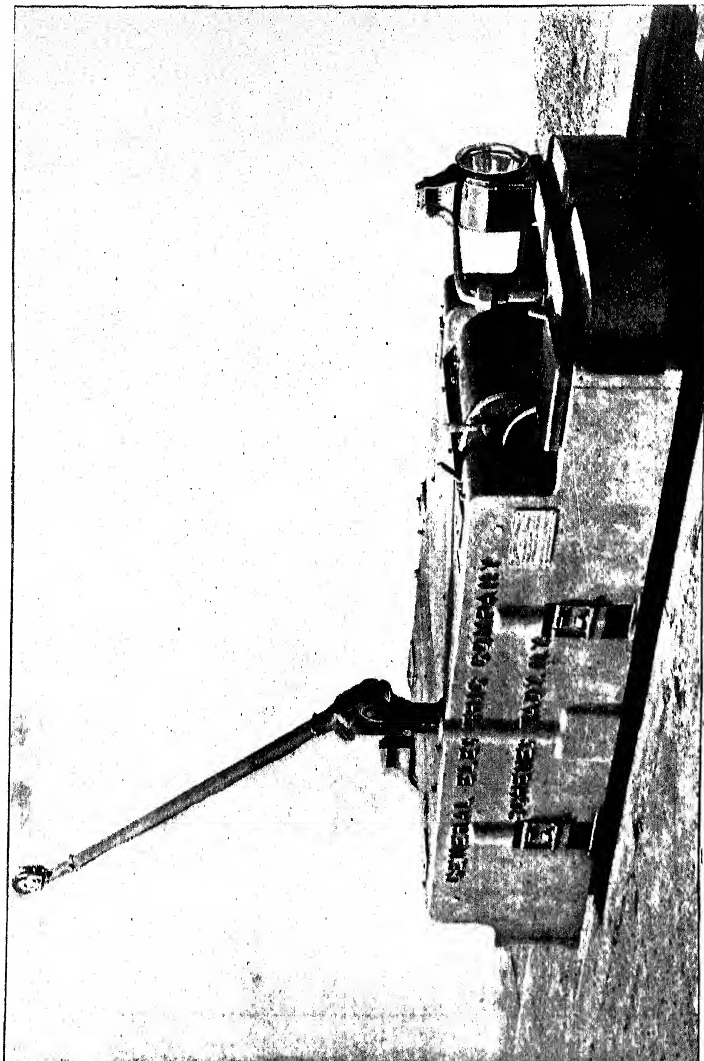


CHAIN COAL-CUTTER.
Equipped with
Three-phase Induction Motor.

Fig. 24.

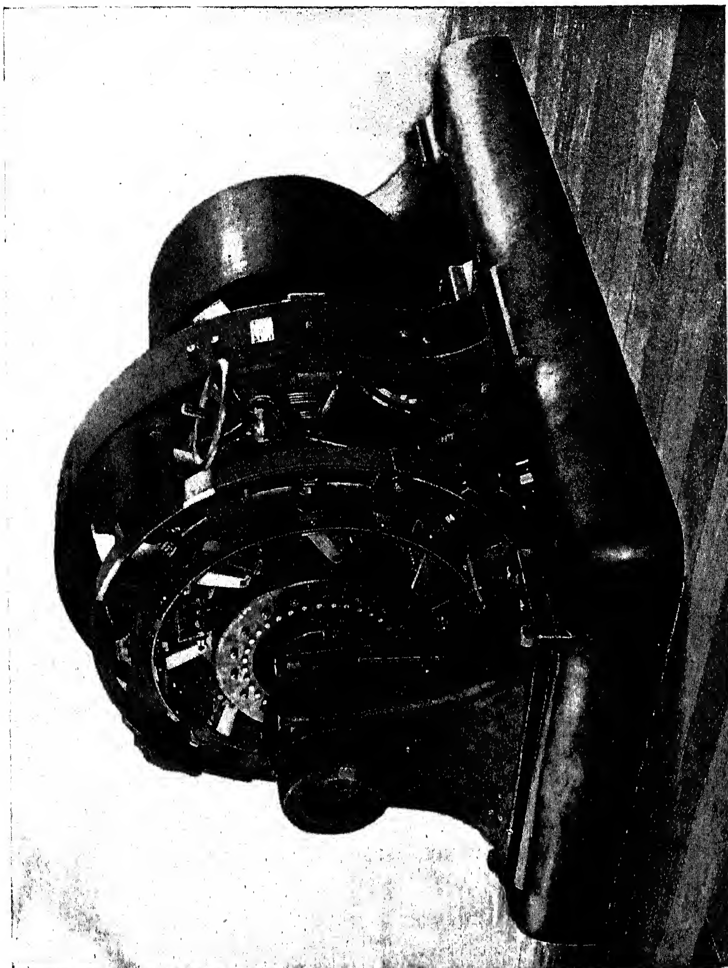
ELECTRIC AUGER COAL-DRILL.

Fig. 25.



ELECTRIC TWO-MOTOR MINING LOCOMOTIVE.
Revenue Tunnel, Ouray, Colorado.

Fig. 26.



ELECTROLYTIC GENERATOR.

Sketch of a Portion of the Gunnison Gold Belt, Including the Vulcan and Mammoth Chimney Mines.

BY ARTHUR LAKES, DENVER, COLO.

(Colorado Meeting, September, 1896.)

WITHIN the past few months I have had occasion to visit and examine the area of country in Gunnison county locally called the Gold Belt and extending from the Cebolla river on the west to the head of Taylor Park and the Sawatch range on the east. The region including the "belt" may be divided topographically into two parts, a northern and a southern one. The former is characterized by lofty mountains and deep cañons, with occasional parks and wide valleys. Most of it is included in the granitic system of the Sawatch range, with local patches of the Palæozoic sedimentary rocks, more or less traversed by great bodies of porphyry and other eruptive rocks. The southern portion, on the other hand, is characterized by comparatively low, rounded hills of schist and schistose gneisses, underlain by coarse, massive granite, which outcrops locally in small, and sometimes in large, patches. These hills also sometimes form table-lands by reason of overflows of lavas, such as andesitic breccia, rhyolite, trachyte and basalt. The leading geological feature is that of a schistose area of width and extent unusual for Colorado. These schists and gneisses are of pre-Cambrian age; but whether assignable to the Algonkian or to some older system has not been determined. The granite which occasionally outcrops from beneath these schists is, I think, of true eruptive character, since its mode of occurrence indicates a once molten, or at least highly viscous or plastic, condition. Thus, we may observe, included in its mass, numerous fragments of schist, almost like a coarse, volcanic breccia. Again, at its immediate contact with the schists, which often stand vertically upon it, the pediments of the latter are fused into the granite, and granite tongues and veins run up amongst the schists, as only a molten or viscous

body could do. The impression I derived was that the schists had been, so to speak, floated up on an underlying molten or semi-molten sea of granite. The granite in question is sometimes a coarse, red granite, sometimes a rather homogeneous, even-grained, gray granite, well adapted for building purposes, and, in fact, selected at Aberdeen quarry as the stone for building the Denver Capitol. Occasionally, but rarely, it is traversed by fine-grained veins of a red pegmatitic granite, not unlike the fine-grained eruptive granite of Cripple Creek and Pike's Peak.

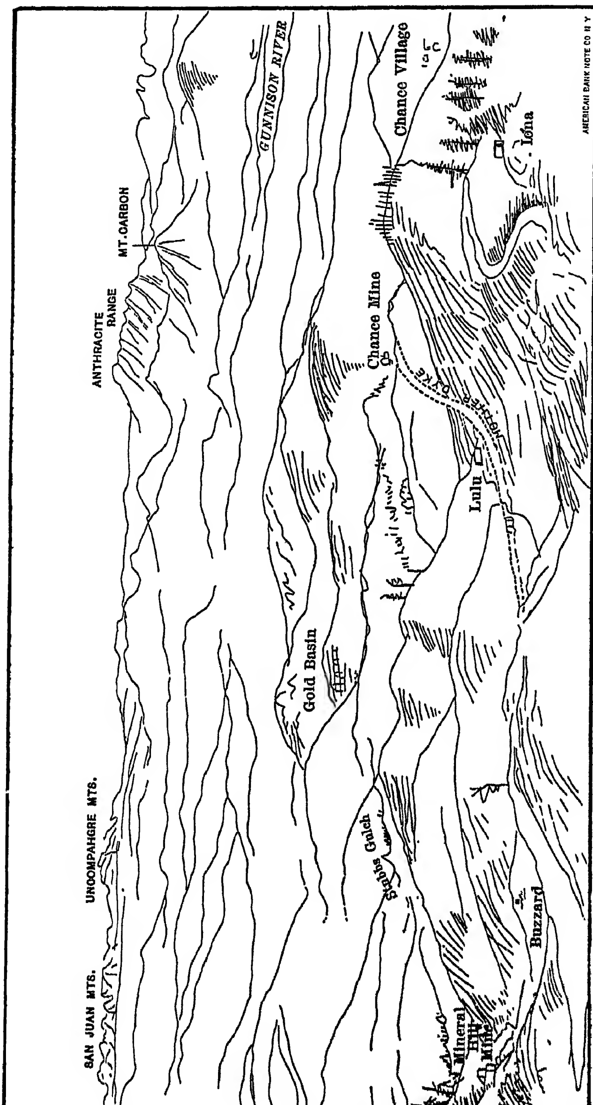
The schists are commonly of dark color, carrying more hornblende than mica. Sometimes, especially in the Vulcan mine region, they are composed of a silvery white mica, forming what are known as sericite schists. Where, by some metamorphic action, the sericites are reduced to a very light tint, the schists assume a beautiful silky texture and color, reminding one of a white asbestos. These are locally called "silver schists" by the miners, and are especially noticeable in the Vulcan mine. The general aspect of the schists is a greenish gray, a greenish tint being often the result of alteration of hornblende. Locally we find them beautifully veined with feldspar and a chalcedonic quartz and jasper, till they resemble certain forms of petrified wood. From a genuine fissile schistose structure the rock may pass into a dense compacted form, which, perhaps, might more strictly be called a gneiss. Generally, by the prevalence of these quartzose veins, both small and great, they suggest the penetrating and crystallizing influence of heated waters, dissolving and redepositing quartz in various forms. Occasionally the veins show signs of contortion.

Of genuine eruptive rocks we have occasional dikes of dark diabase, coming up through the schists, with possibly some basaltic dikes or dark andesites; when they are more or less decomposed, it is difficult to distinguish them at sight from altered, dense hornblendic gneisses. Thus the miner may sometimes think he has a genuine contact-vein between schist and an eruptive rock, when the latter may only be a dense or altered hornblende gneiss.

Resting upon the eroded edges of the schists, and in sections near Gunnison City, on nearly horizontal bodies of Dakota

sandstone, are large bodies of a very coarse andesitic breccia, composed of large blocks of a dark andesite, sometimes more

FIG. 1.



Panoramic View of Part of the Gold Belt and the Distant Elk Mountains, from above Mineral Hill. The Vulcan mine, referred to in this paper, would be located in this bird's-eye view at a point $1\frac{1}{2}$ inches below the top border, and 1 inch to the right of the left border, of the engraving.

than a yard in diameter, together with smaller sub-angular fragments, cemented together loosely by a sand of andesitic

elements. These bodies of breccia are sometimes 400 or 500 feet in thickness and form curious hills cut into grotesque forms, such as the palisades near Gunnison City. They occur in greatest force on the north bank of the Gunnison river and have been little prospected for gold.

Overlying this breccia, and apparently next in order of eruption, is a flow of trachyte or trachytic tuff of a porous character, and sometimes, as near Cebolla, perforated with large and numerous holes, as if from the escape of steam, giving the rock the appearance of a gigantic sponge.

Overlying this, again, as at Cinder Hill, is a flow of basaltic or doleritic lava, of a somewhat recent and fresh appearance, with bands of vesicular scoria.

In the gulch descending from Cinder Hill, near Iola station, we have a section showing the order and succession of the eruptions and their relation to one another. At the base is:

1. Eruptive granite, containing in its mass fragments of schist.
2. Upon this the schists stand vertically.
3. On their eroded edges is a belt of gently-dipping, almost horizontal, Dakota sandstone.
4. On this a bed of andesitic breccia, which is
5. Overlain by a sheet of trachyte.
6. And this by vesicular basalt.

The eruption of these volcanics would appear to have taken place at least after the Dakota Cretaceous and probably towards the Tertiary period.

I must also mention another curious feature which I have not noticed elsewhere in Colorado, namely, a body concerning which I have been in doubt whether to call it an eruptive dike or a great feldspathic vein. I incline towards the former view. It is a body of pink orthoclase feldspar, from 50 to 100 feet in width, very homogeneous and uniform in character and composition, which traverses the schistose country for some 10 or 12 miles in a general N.W. and S.E. direction, like a great red pathway. It is generally composed of quite large crystals of orthoclase, which lie scattered along its path in large and small fragments. This feldspathic body is often minutely reticulated with quartz veinlets, which seem to be of secondary origin. The "mother-dike," as it is locally called, has been prospected here and there, and found to assay fairly as low-grade gold,

and locally to carry pyrite and some copper. It is said (and I think the statement is probable) that it not only cuts through all veins in its path, but also more or less faults them. It is a noticeable feature in a portion of the region from Iris to Mineral Hill, and beyond that to the Lucky Strike mine; and it seems to be the central zone around which, or parallel with which, the principal ore-bearing veins and ore-zones are congregated.

Of the veins in this schistose region some are doubtless small, narrow, lenticular bodies; others are large veins of quartz, or quartz and feldspar, from 5 to 10 feet in width, between well-defined walls, and, generally speaking, so far as my brief observation goes, lying conformably between the strata in dip and strike. These veins appear to be continuous, and are traceable for sometimes several thousand feet in length. The shallow developments of about 100 feet below the surface, give no proof as to their continuity in depth. It may be that they end when they reach the underlying granite, at a depth which varies throughout the region; or they may pass down through the schists into the granite itself.

In addition to these genuine veins, there are bodies of hard purple or blue quartzite, traversing the region in a general N. and S. direction, apparently for long distances, and known locally as "blue veins." I am inclined to think they are rather quartzite members of the schist-strata than true quartz-veins. So far as prospected, they are of low-grade in gold.

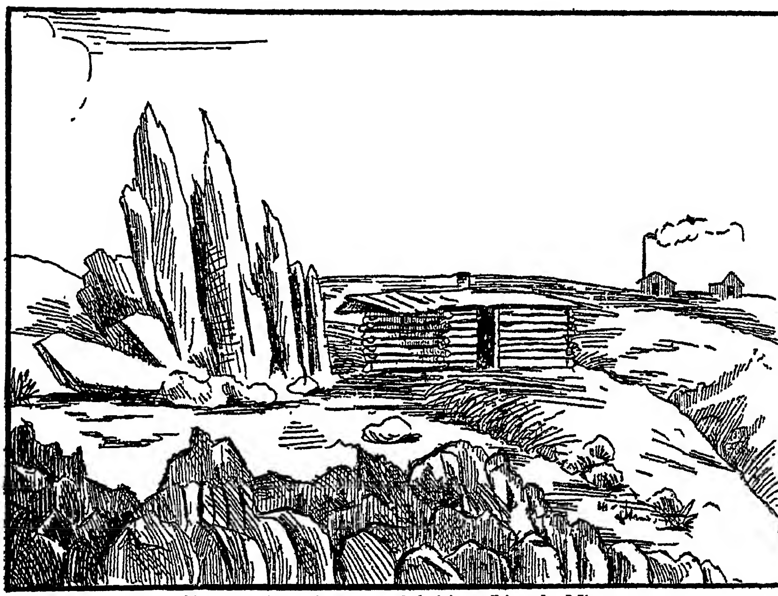
The quartz of the regular veins is a hard gray quartz, often looking very pure, but sometimes impregnated with pyrites. Free gold is rarely visible. The average value of the veins, as shown on the mill-plates, may be from \$20 to \$30 per ton, with occasional larger values. There are, as yet, very few mines that have passed beyond the prospecting-stage. Among these are the Mineral Hill, Old Lot, Lucky Strike, Iron Cap, etc. The rest are mostly prospect-holes.

A peculiar body of fine-grained, red pegmatitic granite traverses the coarser country-granite on the Carpenter property, on Goose creek, reminding one of the eruptive fine-grained granites of Cripple Creek. This body has been cross-cut for 200 feet, and shows pyrites and a certain amount of low-grade gold-ore. Black patches occur in it, as of decomposed hornblende.

From a scientific point of view, the most peculiar and interesting mines of the district are the Vulcan and the Mammoth Chimney, especially as illustrating the geyser or solfataric origin of some mineral veins.

These mines, both located on the same zone, are in the western portion of the district, in an area characterized by sericite schists, whose sharp slab-like outcrops appear on all sides.

FIG. 2.



Characteristic Outcrop of Schists, Lincoln Mine.
(Vulcan and Mammoth Chimney in the Background.)

On Sugar creek, about 5 miles east, a good many eruptive dikes and eruptive masses occur, such as trachyte, rhyolite, andesite and diabase; but I did not observe any striking occurrences of eruptive rock very near the mines in question. On the bank of Camp creek the snow-white dumps and the plant of the Vulcan and Mammoth appear about 100 feet vertically above the bed of the creek. On the road up to the mines we cross the well-defined outcrop which led to the discovery. It consists of a coarse breccia of quartz fragments, both white and rusty, or much honey-combed, mingled with patches of red and yellow jasper, all cemented together by oxide of iron and sand.

The width of this rough outcrop was from 5 to 10 feet. Its limit in length has not been determined; about 100 feet or more, outside of the mining work, is visible. The strike corresponds with that of the sericite-schists inclosing it, viz., E.N.E. and W.S.W.

The dumps of the Vulcan and Mammoth Chimney appear at a distance snow-white; but when we approach them closer we find them streaked with bright sulphur-yellow, red, tawny brown and black. These colors are due to the different material encountered at different depths and thrown successively on the dump. The bulk of this material, especially from the early workings, is a fine white granular quartz-sand, like powdered sugar or salt, mingled with masses of exceedingly light honey-combed, porous, rusty quartz, often showing concentric rings of iron-oxidation. This appears to have been taken from the oxidized zone, very near the surface. Next came large porcelain-like masses of a chocolate brown tint, locally called "brown obsidian," but in reality a brown and banded opal, with conchoidal fracture. With these were duller brown pieces of jasper, veined and ribboned at times with chalcedony and white opal. Specks of fire-opal are occasionally met with, and also fragments of a black flint-like opal, and masses of cream-colored opal, profusely mottled with dark purple or chocolate spots and blotches. All these products, I learn on good authority, assay quite highly in gold, especially the mottled cream-colored, which has shown assays of 16 ounces and also $1100 to $1700 gold per ton. Assayers think the gold is principally in the purple spots. The later deposits, thrown out from the deeper portion of the Vulcan, below 100 feet, are, first, quantities of very pure-looking yellow, granular sulphur; next in order below this large quantities of disintegrated white iron pyrites; and, finally, masses of solid iron pyrites.

Having cursorily examined the dump, we descended first into the workings of the Mammoth Chimney, about 300 feet N.W. from the Vulcan and in the same zone. The shaft is 84 feet deep. At the bottom of this we entered a drift 194 feet long, moving southward towards the Vulcan end-lines. At the end of this main drift cross-cuts, each 26 feet long, have been made on either side, exposing a section of the supposed thickness of the mineralized zone from wall to wall of country rock

(sericite-schist). Down the center of the roof, through the length of the drift, a dark belt extends, composed of somewhat loose breccia or fragmentary masses of dark opal with blackish sand on either side of it. This zone may locally pass into bodies of brown-banded opal or cream-colored mottled opal or into a brown or veined jasper, and, in one place, into a saccharoidal quartz, very much like maple sugar. This brecciated and variegated loose material constitutes the richest part of the ore-bearing zone. No metal of any kind is visible in this upper portion. On either side of this zone, which may be from 5 to 6 feet in width (sometimes more), and forming the walls of the drift, is fine, granular, white quartz-sand, easily gathered by the hand, yet showing withal indistinct lines of former vertical bedding or lamination, revealing its origin from the alteration of the original sericite-schists in place. Mingled with this is at times a fine, silvery-white powder, slippery as a lubricant when rubbed between the fingers, and evidently a relic of the white mica of the original sericite-schist. These walls of sand are at times streaked with red oxide of iron. In the cross-cuts the sand is found to pass into a more compact body of laminated white, opaque, opaline quartz, reminding one of novaculite, only not so hard. From this the zone grades into what is called "silver schist," a beautiful asbestos-like, silver-white schist, and from this into the unaltered greenish-white mica- or sericite-schist of the region.

The section of the zone shown in the cross-cuts may be generalized thus: A central zone of fracture and principal mineralization, composed of brecciated opal, jasper and black sand, and, on either side of this, white quartz-sand, passing into more compact laminated quartz and that into white asbestos-like sericite-schist, with some rusty clay next to the schist country-rock.

The Mammoth Chimney, owing to its shallow depth of 84 feet, had not yet quite reached the sulphur and pyrite zone found in the slightly deeper workings of the adjacent Vulcan. In this we descended by shaft 100 feet to the lowest level. Here much the same phenomena as shown in the Mammoth were repeated, with the difference that, at this depth of 100 feet, the walls on either side the main brecciated, dark, opaline zone were of yellow, granulated sulphur, said to be 15

feet thick, to which, when a match is applied, there is a brilliant blue illumination. We did not observe any pyrite in this sulphur zone; but in the cribbed shaft below the drift, which was inaccessible to us at the time, the sulphur passes down into a quicksand of loose iron pyrites, which kept pouring in on the men who were digging and cribbing the shaft.

The total depth of the shaft is 125 feet, and for the last few feet the loose pyrite has changed into solid, massive, homogeneous, white pyrite.

The sulphur, I understand, assays well in gold; the loose pyrite is nearly barren; and the massive pyrite is of low grade, carrying from $4 to $14 per ton.

This mineralized and altered zone appears to me to have been at one time the seat of solfataric or geyser-like action, similar to that at present in active operation at Steamboat Springs, Nevada, and Sulphur Bank, California.

The hot waters, accompanied by steam and gases, ascended through a line of fissure or weakness in the schists. This fissure may have passed down through the zone of schists into the underlying and semi-molten granite, which may have supplied the heat to the mineral solutions and perhaps the silica of the opal. The heated waters dissolved out silica from the rocks through which they passed and deposited it in a gelatinous condition on either side the line of fissure, forming the brecciated opaline zone. This opal is due to solutions, not, like obsidian, to igneous heat alone. From this central zone the steam and gases penetrating the adjacent schists decomposed them for some width, leaching them of their iron and setting the mica and quartz free as quartz-sand and mica-dust, or, in a less disintegrated state, forming compact schistose quartz. Lesser action may also have produced the asbestiform silver-schist.

The same waters were laden with the elements of iron pyrites, carrying gold, and originally may have reached the upper and now oxidized zone as a body of pyrite, which, near the surface, has been oxidized into the porous, honey-combed quartz we have mentioned. Lower down the pyrite may have been desulphurized, leaving, in place of pyrite, a bed of sulphur; below that the bed of disintegrated pyrite; and finally the unoxidized pyrite-vein, which may continue to depths unknown.

The Smuggler-Union Mines, Telluride, Colorado.

BY J. A. PORTER, DENVER, COLO.

(Colorado Meeting, September, 1896.)

In offering some data relative to this property, and the treatment of its ores, it is proper to say that a purely scientific article has not been attempted. It is hoped, however, that the economic conditions presented may be of interest to the members of the Institute, and that a discussion upon the concentration of the ore may throw some light upon a very unsatisfactory part of the work.

The Smuggler-Union mines are located at the head of the San Miguel river, just south of the water-shed between Ouray and Telluride. The latter town, reached by the Denver and Rio Grande railway, is most picturesquely situated, at an altitude of 8500 feet, in a narrow valley, the lower portion of which exposes only the sedimentary rocks. Along the sides of the valley, large masses of coarse alluvial deposits are left high above the present channel of the stream. Although much placer-work has been done, and some very rich material has been found, the limited extent of the isolated tracts has made placer-mining in this locality generally unprofitable. The valley terminates two miles above Telluride in an amphitheater around which walls rise abruptly several thousand feet. Here the company's mill is situated, being reached by a Bleichert tramway from the mines and also by a spur of the Rio Grande railway from Telluride. At this point, not only are the reddish sandstone and conglomerate beds exposed, but the contact between the conglomerate and the overlying gray volcanic breccia is plainly in view, even at a distance of many miles. A marked stratification is noticeable, indicating the successive layers of igneous breccia, locally termed trachyte. Throughout the entire region this is the favorite name among the miners for the rocks composing the igneous cap of the San Juan

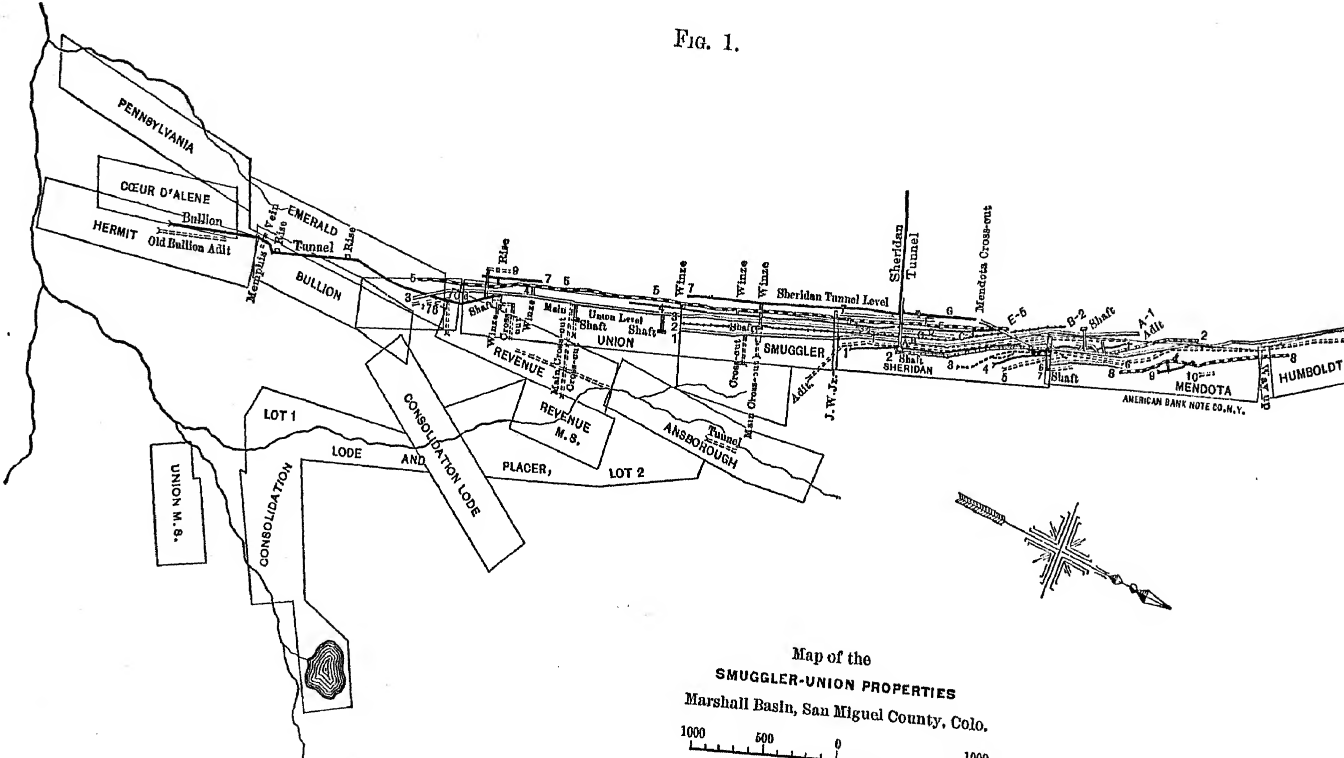
country. I may say in passing that the vague term San Juan, although originally more specific, now seems to comprise all that very high region which is situated on the head-waters of the Rio Grande, Animas, Dolores, and San Miguel rivers, and the Lake fork of the Gunnison. It covers some 50 miles square, mostly on the western slope of the continental divide, and includes the prosperous towns of Durango, Silverton, Rico, Telluride, Ouray, and Lake City. In this area many hundred peaks rise to an elevation of over 13,000 feet.

The geology of the district near Telluride and as far south as Mount Wilson is now receiving the attention of the United States Geological Survey, Mr. Whitman Cross and party having spent several months of the past summer in this field-work. In advance of the government report, Mr. Cross has kindly prepared a paper for the Colorado Scientific Society, which enables me to quote from the highest authority as to the local geology. I select from his very interesting and exhaustive paper only a single page, which briefly describes the formation in which the Smuggler vein occurs. The Sheridan mine referred to by him adjoins the Smuggler, and is one of the properties worked by the Smuggler Company.

"I have especially described the San Miguel conglomerate, a sedimentary deposit of much interest, lying immediately beneath the volcanic rocks of the region, and while containing little volcanic material among its pebbles, it is clear that the San Miguel formation belongs to the general period which witnessed the beginning of the volcanic outburst in southwestern Colorado. The San Miguel conglomerate forms very conspicuous cliffs on both sides of the San Miguel river above Telluride. Upon it rests a stratified, and, as I now believe, waterlaid series of volcanic tuffs and breccias, constituting the lower member of the volcanic complex. This bedded formation, consisting almost entirely of andesitic *débris*, has a thickness varying from a little more than 1000 feet to somewhat more than 2000 feet. Its lower limit is seen on the trail leading up Marshall creek, at an elevation of 9800 feet, and it extends to the level of the Sheridan mine in Marshall basin at an elevation of nearly 12,000 feet. As the beds have a gentle dip, it appears that the thickness of the formation in this section cannot be less than 2000 feet.

"It is proposed to call this stratified series of andesitic tuffs and breccias the San Juan formation, as it clearly plays an important part in the make-up of a large portion of the San Juan mountains. The texture of the San Juan beds varies from a thin-bedded, fine-grained tuff, containing no large fragments, to a tuff-breccia, consisting of large subangular blocks imbedded in a finer-grained matrix. The proportion of large fragments varies a great deal, but I have not seen the formation to be made up of large fragments exclusively at any point, although where it is indurated in proximity to some of the large diorite stocks, and in near mineralized areas, it has the appearance of a massive breccia. The upper limit of the

FIG. 1.



Map of the
SMUGGLER-UNION PROPERTIES
Marshall Basin, San Miguel County, Colo.

San Juan formation may not be always clearly definable. In the mountains about Marshall basin it is, however, very sharply defined by the appearance of the first massive lava-flow of augite-andesite."

The Smuggler vein is remarkable for its continuity and regularity. It crosses the water-shed of Cañon creek, a tributary of the Uncompahgre, and Marshall creek, which runs into the San Miguel at Pandora, at an altitude of 13,200 feet, at which point the thickness of the rocks of igneous origin might be roughly estimated at 3500 feet, since Marshall creek exposes the underlying conglomerate in nearly horizontal position at an altitude slightly less than 10,000 feet. The vein is plainly visible upon the surface, where it crosses the divide, and cuts through the rhyolite and augite-andesite down into the andesite breccia to within only a few hundred feet above the conglomerate, as it crosses Marshall creek. What the character of the vein may be after passing into the conglomerate and sandstone is not yet determined. It will be many years before any of the mines of Marshall basin reach the horizon of the conglomerate.

The accompanying map, Fig. 1, shows the summit of the divide and profile of the vein-crop through the Smuggler-Union property for a distance of one mile. The vein continues south beyond the ground indicated on the map across the Savage fork of the San Miguel, and is plainly visible at the outcrop for over a mile farther in that direction. On the north side of the divide the vein is worked for two claims (3000 feet) in length quite extensively, and can be traced at intervals on the surface for over two miles. The total distance more or less exposed by mining work is more than four miles.

The course throughout the section indicated on Fig. 1 is about N. 20° W., and the dip is westerly at an angle of 75°. The strike throughout the entire workings is very uniform, as will be seen by the parallelism of the levels in plan.

Only two cross-faults occur in the entire length exposed by underground workings. The first is at a point near the south end of the property, and is caused by a large quartz-vein, the Pandora, containing gold, but little or no silver. It dips 45° to the south. The plane of fault is nearly at right angles to the course of the Smuggler vein, and the movement of the fault is about 50 feet. The second fault of only a few feet is made by

the Revenue vein, which crosses the Smuggler vein at an angle of about 15°, and is as well-defined as the latter. This vein is now being developed, and is very easily worked, owing to the decomposed state of the ores it contains. It carries less gold and more lead than the main vein. The course of the Revenue is indicated clearly on the map in plan. The deep tunnel follows this vein for some distance, as a matter of economy in reaching the shaft.

The Smuggler vein is probably a fault-fissure. Both the hanging- and the foot-wall show large polished surfaces. Striation is very frequent, and gouge-matter, several inches in thickness, occurs in places upon the hanging- as well as the foot-wall. Yet, although this is largely the case, long distances occur where the quartz, which almost exclusively forms the filling of the vein, shows no parting whatever from the country-rock. To use a miner's expression, it is frozen to the walls.

The average thickness of the vein is about 5 feet, seldom narrowing to less than 2 feet, and rarely widening to as much as ten. Enclosures of country-rock are frequent, sometimes as fragments, and more frequently in continuous masses between bands of the lode. Cavities are rare; and no such structure occurs as is seen in some veins where corresponding minerals are found at each wall, and others succeed in comformable layers towards the center. In the Mendota claim, a mass of andesite several hundred feet in length and 30 or 40 feet in thickness divides the vein; or, as usually expressed, the lode forks and comes together again in several hundred feet. This is the only point in the workings where such an occurrence is met with, except on an exceedingly small scale, when a stringer leaves the vein for a few feet only. Associated with quartz, rhodochrosite occurs in places and imparts a pinkish color to certain bands in the vein, which are sometimes nearly a foot in thickness and very regular for many yards. When this mineral is present in sufficient quantity to color the vein, that portion is seldom rich in the precious metals. Very small quantities of calcite, brown-spar and heavy spar also occur.

To give an idea of the chemical composition of the gangue, I would call attention to the accompanying analysis made from a monthly sample of the material going to the mill. This is only

to indicate the gangue-matter, and shows how completely quartz predominates. The appended analysis of the concentrates, which still contains nearly one-half gangue, is intended to give some idea of the constitution of the ore.

Battery Sample, April.

Assay.

	Oz. per ton.
Au,	0.53
Ag,	13.10

Analysis.

	Per cent.		Probably Combined as
Insoluble,	80.26		
Al_2O_3 in insol.,	8.52		
SiO_2,	70.42	SiO_2,	70.42
Total Fe,	5.20	FeS_2,	5.57
Fe as Fe_2O_3,	2.55	Fe_2O_3	3.64
Mn,	0.81	$MnCO_3$	1.69
Total Al_2O_3,	9.94	Al_2O_3,	9.94
CaO,	1.98	$CaCO_3$,	3.54
Mg,	0.28	$MgCO_3$	0.98
Zn,	1.02	ZnS,	1.52
Pb,	0.66	PbS,	0.76
Cu,	0.06	Cu_2S, Fe_2S_3	0.17
S,	3.77		
Alkalies undetermined.			98.23

Screen-Analysis.

Mesh.	Per cent.	Au. Oz. per ton.	Ag. Oz. per ton.
On 20,	0.02		
" 40,	18.17	0.25	7.68
" 60,	18.93	0.66	8.33
" 80,	9.40	0.83	10.17
" 100,	4.93	1.20	10.90
" 120,	4.96	0.93	11.07
" 150,	9.04	1.13	13.53
Over 150,	34.55	0.51	19.64

NOTE.—No tellurium or antimony was found.

Concentrates, April.—Lots 197 to 259.

Assay.

	Oz. per ton.
Au,	2.78
Ag,	50.43

Analysis.

			Probably Combined as
Insoluble,	46.26		
Al_2O_3 in insol.,	4.84		
SiO_2,	39.92	SiO_2,	39.92
Total Fe,	15.85	FeS_2,	26.89
Fe as Fe_2O_3,	3.15	Fe_2O_3,	4.50
Mn,	1.68	$MnCO_3$,	3.51
Al_2O_3,	6.20	Al_2O_3,	6.20
CaO,	2.51	$CaCO_3$,	4.48
Mg,	0.65	$MgCO_3$	2.27
Ba,	0.00		
Zn,	3.98		
Pb,	3.26	ZnS,	5.94
Cu,	0.18	PbS,	3.76
As,	0.34	Cu_2S, Fe_2S_3	0.51
Sb,	0.00	As?	0.34
S,	16.82		
Alkalies undetermined.			98.32

Screen-Analysis.

Regular sample before grinding of Co. lot 339. Car No. 1884.

Mesh.	Per cent.	Au. Oz. per ton.	Ag. Oz. per ton.
On 20,	0.18		
" 40,	9.53	2.13	24.53
" 60,	6.95	4.35	41.90
" 80,	4.83	5.66	44.17
" 100,	3.35	5.06	44.27
" 120,	5.34	3.53	41.80
" 150,	9.90	2.53	40.80
Over 150,	59.92	1.26	61.07

Assay-Sample, Lot 339.

	Oz. per ton.
Au,	2.40
Ag,	51.72

I am indebted to Mr. F. Roeser, Chemist of the Omaha and Grant Smelting Works at Durango, for the above analyses.

The minerals occurring are pyrite, chalcopyrite, galenite, sphalerite and the arsenical silver-minerals. Proustite and polybasite have been determined, and probably nearly all of the arsenical silver-minerals occur. No specimens of tetrahedrite have been recognized. Metallic silver is very rarely encountered. Metallic gold is more frequent, although unusual. The remarkable feature of the vein is the regular and continuous distribution of the ore, which generally lies near the foot-wall. The usual occurrence is a few inches of richer ore (so-called

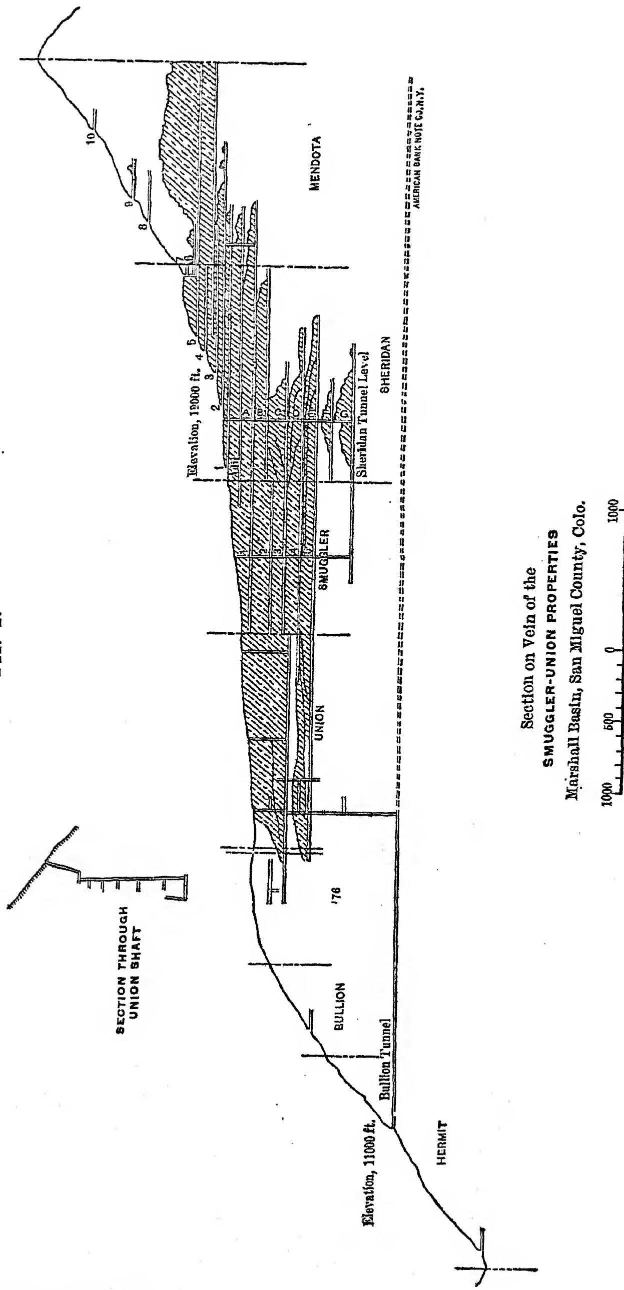
"shipping-ore") and a foot or two of banded structure, more available for concentrating, which goes to the mill. So constant is the occurrence that over a mile has been worked along the vein on various levels without meeting any part where the ore is not present in sufficient quantity for continuous stoping. One remarkable feature is the constant increase of gold-value toward the south, throughout the entire workings, and the corresponding decrease in silver-value in same direction. Where the vein crosses the divide, at the extreme north end of the property, the gold-value is hardly one-quarter that of the silver. The transition from silver to gold is almost constant, until, a mile south, the vein practically becomes a gold-lode. This change does not correspond to depth gained by slope of mountain toward the south end of the property; for on the north side, at a corresponding altitude, the vein contains gold to the extent of only a few per cent. of its total value.

Fig. 2 exhibits the underground workings in section on the plane of the vein. The shaded portions have been stoped out. Dotted lines indicate the projected continuation of the Bullion tunnel.

The levels on the vein have been run heretofore 100 feet apart, and are timbered with stulls, upon which lagging, to support the waste produced in stoping, is placed. The stull-timbers are set into foot- and hanging-wall at right-angles to the dip, and are about 14 feet long and as many inches in diameter. The work of exploitation is carried on by stripping in country-rock on the hanging-wall to some 7 feet in height, and then shooting down the vein in mass, to be broken and passed into chutes with little or no sorting.

An important change has recently been introduced in the construction of these chutes, which has made it possible to use them in spite of the passage of heavy and sharp rock, until a height of even 250 feet has been attained; thus saving the great expense of running and timbering drifts at every hundred feet. The levels will in future be 200 feet and more apart. The change in construction is locally a new departure, and consists in simply placing the timbers used in building of chutes (mills) so that the end will be cut into, instead of the sides, which are soon cut out. The old "mills" were built like a log-house; the new are constructed of short mill-timbers, and

FIG. 2.


MENDOTA
SHERIDAN
Sheridan Tunnel Level
SMUGGLER
UNION
'78
BULLION
Bullion Tunnel
HERMIT
Elevation, 12000 ft.
Elevation, 11000 ft.
SECTION THROUGH UNION SHAFT
AMERICAN BANK NOTE CO.,N.Y.
Section on Vein of the
SMUGGLER-UNION PROPERTIES
Marshall Basin, San Miguel County, Colo.
1000 500 0 1000
Scale of Feet

for the first hundred feet are built with ends presented to the falling rock.

From the chutes ore is hauled by mules to the shaft, in cars containing about 1 ton each. From the collar of the shaft, which starts on the third level, the distance down to the tunnel is 700 feet; and all ore is lowered to that level. From the bottom of the shaft a train of twelve cars is hauled to the sorting-house at the head of the tramway. A very rough hand-sorting here takes place while the ore is being filled out of the chutes over aprons into cars, which load into tram-bins. About 10 to 15 per cent. of high-grade ore is thus selected for shipping, and goes over the tram direct to the railroad and then to the smelter.

From the sorting-house the lower-grade ore is transported over a Bleichert tramway, at a cost of about 25 cents per ton, to the mill. The tramway is 1 mile in length; the alignment being straight and the inclination about 20°. Fourteen towers support the standing cable, the longest span of which is 1200 feet. The traction-cable is $\frac{7}{8}$-inch and the standing-cable $1\frac{3}{4}$ inches in diameter. The buckets used are 38 in number and contain 500 pounds each. The monthly capacity of the mill is 5000 tons; and this amount the tramway delivers in ten hours daily, which is far above the guaranty of the makers. The total cost of the wire tramway was $30,000. It is only fair to say that it has given the greatest satisfaction, and cannot be too highly recommended for the transportation of ore, particularly in mountain regions, where other means are difficult and expensive.

Upon arrival at the mill, the ore is broken to a suitable size through a Blake crusher, and is thence carried by automatic feeder to stamps, which crush to 14-mesh. This entire product is passed over Triumph and Frué tables, producing about 15 per cent. of concentrates, which contain an excess of silica, as the analysis already given shows.

No farther method of concentration is used, except that a rough sizing of the tailings from the vanners is made; the coarser part, which will not pass through a 40-mesh screen, being reground in a Huntington mill, and then retreated on a second series of vanners. A detailed description of the mill would be uninteresting. The stamps and mortars used are of

similar construction to those in general use for gold-milling on the Pacific coast. The stamps are 800 pounds in weight, and drop 8 inches, with 95 drops to the minute. The 50 stamps crush 150 tons daily.

The above seems to be a very crude process of concentration; and the resulting loss is startling, being nearly 20 per cent. of the gold and fully 40 per cent. of the silver contained. The average contents of the milling-ore for the past year has been slightly over ½-ounce gold and 12 ounces silver per ton. The minerals associated with the precious metals are so disseminated in the gangue that, unless very fine crushing is resorted to, little concentrating material is liberated. Preliminary experiments have been made with a view to reducing the ore to proper size by rolls, thus avoiding an excessive quantity of finely divided material. (See screen-analysis given above.) A crushing to wheat-size permitted a concentration in jigs, showing very rich headings; but even with crushing to this size, only a small percentage of the precious metals was obtained by jigs. It was found necessary to recrush the resulting tailings and middlings very fine, in order to liberate the minutest particles of ore contained. Hence from an economic point of view, it has not seemed advisable to introduce rolls and jigs, in connection with stamps, to effect so small an additional saving.

During the past few years, transportation by pack-train has given place to the tramway, thus saving many dollars per ton. The ore is delivered from the mouth of the tunnel, opened far below the deepest working. Operations in the mines are conducted on a large and economical scale; but the startling loss by concentration still goes on, and over 30 per cent. of the precious metals goes daily down the San Miguel river. No important progress is being made in this direction, to correspond with the great saving effected in other branches of the business.

Faulting and Accompanying Features Observed in Glacial Gravel and Sand in Southern Michigan.

BY CARL HENRICH, NOBLE, ILL.

(Colorado Meeting, September, 1896.)

In the winter of 1895 to 1896, during the construction of the Jackson and Cincinnati railroad, running from Addison, Lenawee county, Mich., to Jackson, Mich., in a northerly direction, I had occasion to observe some interesting geological features, described below, in a cut of this railroad made through the low ridge of gravel and sand, dividing the valley of Silver Lake and Silver Creek from the Goose Lake and Goose Creek valley, in Lenawee county.

The surface-deposits of this part of the county are, as is well known, of glacial origin. Gently rounded, long ridges of sand, gravel and clay intervene between the marshy bottoms and the lakes of the valleys. The nature of the deposits, varying from a sandy clay or clayey sand to fine and coarse gravel and pure siliceous sand on the one hand, and to a very tough, fine, uniform and often indurated clay on the other, as well as the irregular and sudden changes, sharply divided, between these materials composing the ridges, in the cuts made through them by the railroad, would admit of no doubt that this country owes its present surface-configuration primarily to the action of the last glacial period, which covered its surface with an immense ice-sheet. The well-known ice-markings on the huge granite, schist, greenstone and conglomerate boulders found everywhere imbedded and protruding from the surface, add their testimony to the other characteristics mentioned.

Most of the cuts made by the railway through the ridges showed in a marked degree the irregular arrangement and the sudden changes of and between the materials mentioned above.

One cut, however, known as Kelly's cut, or Kelly's gravel-pit, about 4½ miles north of Addison, and about 1½ miles south

of Kelly's Corners, exhibited an almost horizontal arrangement of layers of finer or coarser gravel, alternating with layers of fine, siliceous sand. This regular horizontal division or stratification of the materials would in itself have been remarkable, as it formed a notable exception to the irregular arrangement of the various materials composing the ridges, exhibited by the cuts of the railway farther south. But still more remarkable appeared a regular and well-defined system of fissures, which, under angles of approximately 60° to 70° from the horizontal, and in two or three directions, traversed the horizontally stratified sand- and gravel-layers, and everywhere faulted these layers. There are two well-defined systems of these parallel fault-planes, which intersect each other, probably under an angle of about 60° in the horizontal plane.

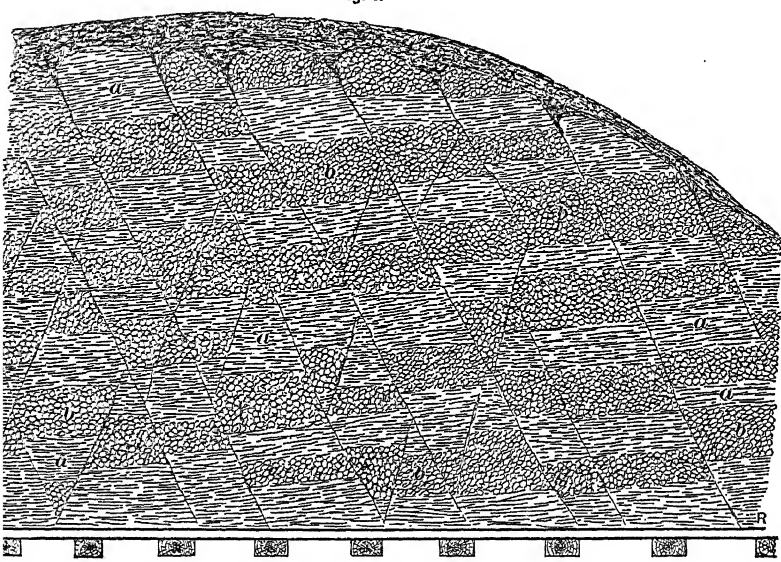
The alternating deposits of gravel and fine sand in horizontal layers bring out in a very marked manner the vertical throws caused by these faults. These parallel fault-planes are from 5 to 12 feet apart; and I was unable to find a single one which did not show a very marked throw, of certainly not less than 7 inches, of the gravel- and sand-layers on either side. The multitude of the fault-planes; the presence of at least two, and probably three, systems of them, intersecting each other not only horizontally, but also vertically; the comparative thinness and frequent alternation of the sand- and gravel-layers, and the uniform color and similarity of the various sand-layers rendered it impracticable to measure precisely the actual throw caused by each fault-plane; but it may be positively stated that in no case was any throw observed which would be less than 7 inches vertically.

Besides throwing and faulting the layers of sand and gravel, these systems of intersecting fissures were further characterized and brought out prominently by carrying, if not being actually filled with, nodules and ramifying tubules. Each of these contained a central cavity, around which a whitish powdery substance gradually but quickly passed into a reddish, ferruginous, cemented outer shell. The siliceous sand seemed to have been decomposed and powdered by the moisture or solution evidently circulating at times through the central cavity. The occurrence of these nodules and ramifying tubules recalled very vividly the characteristics and descriptions of the "*Loesskindchen*," and

their disposition to form in apparent bedding-planes (in this case replaced by undoubted fault-planes).

The question arises: Is the water circulation, which exists undoubtedly at times in this system of fault-fissures, and which formed or forms these nodules and tubules, an ascending, or a descending one? While I was not in a position to make any chemical analysis of the constituent parts of these tubules, I judged from the evident decomposition of the powdered rem-

Fig. I.



Section of Kelly's Sand and Gravel Cut.
R R, railroad. *a a*, sand. *b b*, gravel.

nants of siliceous sand in the interior of the nodules, and the probable calcareous and ferruginous character of the cementing material of the outer shell, which binds the sand grains together into the tubule, that the water or solution which must be considered as the forming cause of these tubes, most likely came from below, where the underlying Permo-carboniferous or Devonian rocks of this region might supply the lime and iron which the rain-water percolating through a foot or two of the overlying soil would hardly be liable to bring with it, for the formation of these tubules filling nearly the whole extent of all these numerous fault-fissures.

The accompanying figure, from a sketch made by me in April, 1896, shows the appearance of the fault-systems on the east side of the cut.

The sand and gravel composing the ridge is overlain by a sandy, clayey loam of reddish color and variable thickness. It is on an average probably about 2 feet thick, but at the outcrops of the fault-fissures, and especially where the intersection of two fissures happens to come to or near the surface, the decomposition of the sand and gravel into loam reaches in places a depth or thickness of 4 and 5 feet, widening the fissures at their apexes.

The question naturally arises: What caused these extensive and well-developed systems of fault-fissures in the loose material, not yet solidified or cemented into rock, and never overlain by any considerable thickness of other material from the time of its original deposition in the glacial epoch to the present day?

The explanation of the cause of these systems of fault-fissures, which occurred to me as the most plausible, is that the two masses of ice, or glaciers, which in converging directions scooped out the valleys of Silver Creek and of Goose Creek, exerted from both sides on the intervening ridge of sand and gravel a lateral pressure nearly horizontal, or possibly slightly downwards towards the axis of the ridge from both flanks. This horizontal (or nearly horizontal) pressure from both sides, exerted on the stratified sand- and gravel-layers of the ridge (which at the same time may have been covered by a thinner sheet of ice), may have resulted in the development of these intersecting systems of parallel fault-fissures, their angle of intersection being approximately the same as the angle of intersection of the two glaciers exerting the pressure.

This occurrence illustrates, further, that the systems of fault-fissures which we find in stratified rocks may have been formed, and even the vein materials filling such fissures may have been deposited, before the materials forming these stratified rocks had been really solidified or cemented into rock, and before these rocks were overlaid by other rock-forming sediments. It suggests, also, a very natural explanation of a hitherto puzzling phenomenon, where a faulted stratified rock or series of such faulted strata of rocks are found between non-faulted masses or

strata. Moreover, it would furnish a natural explanation of the occurrence in certain stratified rocks of veins or systems of veins and veinlets (often intersecting and called "stockwerks") which cut off sharply at the underlying or overlying rock-contact, or at both. In fact, it occurred to me that this simple occurrence, accidentally observed, might be useful in explaining some hitherto puzzling problems in faulting and in ore deposits.

The cut is still in existence, and liable to remain in evidence for considerable time. I should have liked to get a good photographic picture of the east side of the cut shortly after its steep slope had been excavated, and when the strong prevailing winds, combined with dry weather, had brought out all the features described above in a clearer and better way than the cut will be likely to exhibit, now that the leveling and obliterating agencies of rain and the forming of the natural permanent slope have had time to destroy most of the then strikingly clear characteristics of the occurrence. Still, the place will be always accessible for the verification of these features by any subsequent investigator.

Further Notes on the Alabama and Georgia Gold-Fields.

BY WILLIAM M. BREWER, ATLANTA, GA.

(Colorado Meeting, September, 1896.)

SINCE writing the paper on this subject,* which was presented at the Atlanta meeting in October, 1895, I have had opportunity for more thorough investigations in several localities, and venture, therefore, to offer some additional facts and opinions.

The literature of the subject is not very abundant. Dr. George F. Becker's "Reconnaissance of the Gold-Fields of the Southern Appalachians"† contains some brief notes relating to Georgia, but does not include Alabama. The paper of Messrs. Nitze and Wilkens on "Gold-Mining in the Southern Appala-

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* *Trans.*, xxv., 569.

† *U. S. Geological Survey*, Sixteenth Ann. Rep., Part III., page 251.

chian States,"* presented at the Atlanta meeting, refers briefly to occurrences of gold-bearing ore in these two States. Two bulletins have been issued, in 1892 and 1896, by the Alabama Geological Survey, in which every known occurrence of gold-bearing ore in that State is briefly described, and the results of microscopic examination of the rocks of the gold region are given. The foregoing publications, together with letters printed from time to time in the *Engineering and Mining Journal*, comprise nearly all the reliable information hitherto published concerning these gold-fields.

Investigations made during the winter and spring of 1896, in both Georgia and Alabama, lead me to believe that, so far as shallow prospecting is concerned, the gold-fields have been thoroughly explored. I doubt whether many new discoveries remain to be made. But further development of those already known may demonstrate in many instances a value not now clearly evident.

Our knowledge of the country-rocks inclosing the auriferous bodies has been greatly increased by the work of the State geological surveys. The microscope has shown that in both Georgia and Alabama the igneous or eruptive rocks are much more abundant than was formerly supposed. In Alabama, however, the gold is not as closely associated with these rocks as in Georgia. On the contrary, the Alabama gold-bearing ores and gravels are more numerous in the belts of semi-crystalline slates than in the fully crystalline schists. Yet the richest "saprolite"-diggings† in the State are in a belt of diorite, closely associated with a green schist, locally known as the Millerville, Iwana or Hillabee green schist, which, as indicated by microscopic examination, is probably an altered eruptive. A belt of this schist forms the northwestern border of the fully crystalline schists, and is continuous across nearly the whole section of the Alabama gold-region.

In the immediate vicinity of Arbacoochee, where from forty to fifty years ago most of the Alabama placer-gold was produced, extensive, though shallow, prospecting has been carried on during the past year. On a tract of 200 acres, embracing low ground traversed by two or three small creeks, an attempt

* *Trans.*, xxv., 661.

† Dr. Becker's nomenclature.

at hydraulic mining was made last spring. Two serious obstacles to this method were encountered: the lack of dumping-ground and the small available supply of water. To overcome the first the Crandall elevator* was employed to handle the tailings. The lack of quantity and pressure of water was not so easily remedied, and I am informed that this was the reason for the suspension of the work, which occurred before the actual value of the diggings had been ascertained by it. Prospecting-pits, however, had indicated that, although the ground had been worked half a century ago with sluice-boxes, rockers and long-toms, there remained enough gold to repay hydraulic mining if sufficient amount and pressure of water could be obtained for removing the overlying earth.

But the prospecting in this locality which was most interesting to me was carried on in search of ore-bodies. From time to time pockets or kidneys of very rich quartz were discovered, the specimens often showing native gold in crystals from the size of wheat-grains up to, and even beyond, that of cherry-stones. But these deposits were so irregular in respect of continuity on the line of the strike as to indicate the probability that mining would prove expensive and unprofitable. Those which have been exposed in the surface-pits are good examples of the "lenticular stringers" described by Dr. Becker, and called by him "linked veins." The discovery in 1895 of one of these rich pockets led to the active prospecting which is still in progress, and the result of which, as depth is attained, cannot be inferred from the present exposure. It is a general characteristic of this formation in Georgia and Alabama that the "linked veins" are narrow, rarely exceeding 18 or 20 inches in thickness. This is also noticeable in the ore-bodies of the semi-crystalline slate extending across the Alabama gold-region, and, as already observed, bounding the crystalline schists on the northwest.

Whether any of these deposits can be mined with profit on a commercial scale has not yet been settled by actual experience. In my judgment there is a much better chance of success in mining the more extensive bodies of lower grade which are found paralleling these formations to the southeast.

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* See Mr. Crandall's paper, "The Hydraulic Elevator at the Chestatee Mine, Georgia," page 62 of the present volume.

The green schists are described at some length by Dr. Eugene A. Smith, State Geologist of Alabama, in a bulletin, recently published, on the upper gold-belt. The peculiar structure and other characteristics of these schists have rendered them highly interesting to all geologists who have examined the crystalline area. With regard to the outcrop in Alabama, Dr. Smith says:

"It may be followed practically without interruption, but with somewhat varying width of outcrop, from Chilton county through Coosa, where it underlies a tolerably wide expanse of country about Stewartsville and Iwana; thence through Clay, about Hollins, Pine Grove, Mountain Meadow, Brownsville, Millerville or Hillabee, Coleta, Chandler's Springs, Fish Head Valley, and on through Cleburne, by Chulafinnee, Arbacoochee, Anna Howe, etc."

In the same report, Dr. Smith says that microscopic examination has shown these rocks to be composed in the main of actinolite, epidote and chlorite, with some quartz, and that they have been designated by Dr. Clements and Mr. Brooks, who made the examinations, as chlorite-epidote schists, actinolite-epidote schists, or chlorite-schists, according to the predominant constituents. In general, they have been altered to such a degree that it is difficult to determine the original rock from which they have been derived; but all experts agree in believing this to have been a basic eruptive.

According to my observation, the ore-bodies of greatest extent, though of low grade, are usually found connected with the mica-schist formation, or with a garnetiferous semi-crystalline slate. It is principally in the belts of these rocks that the present gold-mining of Georgia and Alabama is carried on.

As regards the persistence of the ore-bearing veins or zones in depth, my own investigations lead me to the opinion that their continuity in that direction will be generally found to obey the law almost universally exhibited along the strike of the veins, namely, the interruption of absolute continuity by the divisions due to "lenticular-stringer" structure. Complete proof of this proposition cannot be furnished, because mine-workings of sufficient depth are not yet sufficiently numerous to furnish evidence of a law universal throughout the formations under consideration. I can only say, therefore, that in single instances I have found the lenticular form of the ore-bodies to be exhibited in the vertical as well as the horizontal section. In other words, they were not "chimneys" of lenticu-

lar section, extending indefinitely downward on a given pitch, but more nearly true lenses. In one case, at least, I have found that the two dimensions of length on the strike and length on the pitch were about equal, and that a lens, after reaching a maximum thickness in its downward course, gradually dwindled to very narrow proportions, and finally gave out altogether.

But the other and compensating characteristic shown along the strike seems also to obtain in the vertical section, namely, the disappearance of one lens is accompanied or shortly followed by the beginning of another, standing *en echelon*, within the same zone. In the case just referred to, I found that as one lens died out towards the foot-wall, another of about equal proportions appeared near the hanging-wall. If such a rule could be demonstrated to be universally applicable to the southern auriferous ore-bodies, it would be obviously a great help in our mining.

At the Creighton mine in Georgia, where work has been carried to more than 400 feet vertical depth, the above law has been shown to obtain.

In sinking an incline, 80 feet deep, on the dip of an ore-body in Randolph county, Ala., I found a similar state of facts. At the top of the incline the ore-body was 13 feet thick, and a cross-cut at 45 feet depth showed that this thickness had been maintained so far, but in sinking deeper we found it to diminish, dying out towards the foot-wall. At the same time, the wedge-like top of another body appeared near the hanging-wall, and this body increased until, at the depth of 80 feet, it equalled in thickness the maximum thickness of the one which had disappeared.

Dr. Becker, in his monograph, already cited, devotes much space to the structure of the Southern ore-bodies; and in the recent *Bulletin* of the Alabama Geological Survey, Dr. Smith, remarking that Dr. Becker's conclusions embody modern views concerning rock-structure which appear to be applicable to the Alabama rocks, gives the following condensed abstract of them:

"In Alabama, as well as in the Southeastern States, the structure of almost the entire mass of the older rocks is schistose, the strike of the planes of cleavage being generally that of the Appalachian range, or N. 30° to 50° E. Where sedimentary strata occur they too strike in approximately this direction, in consequence of the folding which has built up the range; and as a consequence of this fact, the cleavage of the schists has very often been mistaken for bedding. The

dip of these schists is generally toward the southeast, though occasionally northwest, as well as other abnormal dips occur. These remarks apply to the more prominent surfaces of schistose cleavage, but observation will generally show that there are schistose partings at a large angle to the most pronounced one, and when the two are about equally developed, their intersection being nearly horizontal, the strikes of each are approximately the same. Occasionally there are other schistose cleavages striking nearly at right angles with the predominant cleavage, but these play a comparatively unimportant part. According to modern views, massive igneous rocks, such as granite, as well as stratified rocks, are converted into schists by the development of planes of cleavage by shearing, *i.e.*, 'by the sliding of each of the parallel infinitely thin lamellæ, of which the rock may be considered to be made up, upon that next below it, in the same direction and by the same infinitesmal amount,' the movement being distributed over an infinite number of surfaces and not reaching the rupturing strain on any one. If the rock is not very uniform in composition, some portions of it may acquire the schistose structure before others, and the resulting schist will show bands or sheets in which different degrees of schistosity will be exhibited. If the deforming movement be carried far enough, fissures will be opened in part of the schist, and caruneulated surfaces, or puckered surfaces, will result, whilst other portions of the schist will have merely flat cleavage-surfaces. It is in such unevenly deformed rocks that fissures have opened most widely and veins are most abundant and widest. 'Carunculated surfaces are therefore properly regarded as favorable indications by the miners.' "

In this connection Dr. Smith goes on to say:

"Prof. Becker finds proof that the opening of the fissures now filled with ore took place later than the movements which rendered the country schistose, in the fact, among other things, that angular fragments of schist are often enclosed in the quartz; on the other hand, the connection of the ore-deposits with dikes does not appear to be very close, but coeval on structural grounds, since neither the ore-deposits nor the dikes have been greatly disturbed since their formation. The observations of Prof. Becker show that in the great majority of cases the fissures follow the schistose partings somewhat closely though not accurately, showing a certain degree of correspondence between the producing forces of two. Nevertheless, the movements were not in general identical in direction. This is shown first, in the existence of veins and fissures at various angles to the schistose surfaces, and secondly, in the circumstance that some of the markings produced in the dislocations accompanying the opening of the fissures do not coincide in direction with the motion which produced the schistosity. Again, where the rocks are schistose and the main ore bodies are intercalated, stringers almost always cut into the walls. From the relation of these stringers to the planes of cleavage of the schists it is inferred with a reasonable degree of certainty that the opening of the fissures has been accomplished by *normal* faulting, while the cleavage of the schists has been the result of *overthrust* or reverse faulting; in other words, the movements producing the schistosity and those producing the fissures have been in opposite direction though approximately along the same planes."

So far as my own observations go they confirm Dr. Becker's conclusions as applicable to a very large extent in Alabama.

Of course, the two main questions affecting gold-mining in the South are: 1. Whether, in the large, low-grade bodies, the

ore is sufficiently rich; and 2. Whether, in the small, higher-grade bodies, the ore is sufficiently abundant for profitable mining.

As to the first question, there are no mines now in operation in Georgia or Alabama from which definite conclusions can be drawn. But preparations are in progress at two or three localities to mine, on a limited scale, extensive low-grade deposits, and I hope to be able to present to the Institute at some future meeting a paper that shall give a report of the progress made in these enterprises.*

As to the second question, we have in Georgia two examples of successful mining upon the narrower but richer ore-bodies. The first is the Creighton mine, which has been described at considerable length in Dr. Becker's monograph and in the paper of Messrs. Nitze and Wilkens. The second is the Walker mine, situated in McDuffy county, about 31 miles northwest from Augusta, and owned and operated by Mrs. J. Sep Smith, who has been conducting it with profit for the past eight or ten years. This is briefly mentioned by Messrs. Nitze and Wilkens. I have referred to both of these operations in my former paper, already cited.

The number of deposits of auriferous gravel and quartz-veins in Georgia and Alabama is very large. In my recently published preliminary report on the upper gold-belt of Alabama I have mentioned at some length each occurrence of gold-bearing quartz in that belt in the counties of Cleburne, Randolph, Clay, Talladega, Elmore, Coosa and Tallapoosa. Dr. Phillips, in his report on the lower gold-belt, published in 1892, described each occurrence in the counties of Elmore, Coosa and Tallapoosa, located south and southeast of the region embraced in my examination.

For convenience we both divide the gold-region under consideration into belts, in each of which we include all occurrences within a given zone of country-rock. Thus the Silver Hill belt, for instance, embraces all the discoveries in a zone of semi-crystalline slate, extending from near the southwestern extrem-

* The success of the Haile mine, in South Carolina, as an operation based on low-grade bodies, furnishes a standard as to what can be done in the way of cheap mining and reduction on a large scale. It has been so abundantly described in our *Transactions* (and elsewhere) that I need not give particulars here.

ity of the crystalline area known as the gold-region, across the State in a northeasterly direction into Georgia and continuing in the same direction for some distance in that State.

Northwest of this belt, separated from it by a belt of gneiss, and only a few miles distant, I found another belt of semi-crystalline slate, or rather the northeastern extension of a belt of this formation, which was referred to by Dr. Phillips as belonging to the lower gold-belt; this we have both designated as the "Goldville and Hog Mountain." It is continuous across a portion of Tallapoosa, Chambers and Randolph counties in Alabama, and from the latter enters the State of Georgia. Some little difference can be observed between these two belts of formation. The Silver Hill slate is not impregnated with the pseudomorphs of limonite after pyrite, commonly called "iron garnets," while the slate in the Goldville belt is associated with immense quantities of "iron garnets," and is followed in the series towards the northwest by extensive granite flat rocks and massive beds of gneiss.

The next belt, which is auriferous, is located a few miles further to the northwest. In this the fully crystalline schists prevail, including mica, hornblende, diorite and the "Millerville, Iwana or Hillabee," already mentioned in this paper.

In considering the gold-regions of Georgia and Alabama from a commercial standpoint, the statistics which have been gathered from time to time in the past have been anything but satisfactory. It is hardly fair to base a final judgment on these statistics. The general rule is that the ore is very refractory. Much of it is so closely associated with graphitic slate and graphite that it is economically impracticable to mine it and separate it from the graphite before sending it to the mill. This graphite is usually in the nature of graphitic slate, in which the lenses of auriferous quartz occur, and through which much free gold has been disseminated—in fact the slate shows by panning about the same value as the quartz. Other ore, which otherwise would be considered free-milling, has a talcoid micaceous schist closely associated with it, and a large proportion of the gold is disseminated through the schist. The structure of the ore-body in these instances is such that it cannot be profitably mined if sorting is attempted. In many of the ore-bodies a considerable quantity of wad or bog-man-

ganese is associated with the ore. But the greatest amount of the refractory ore is rendered so because of the iron pyrites with which the gold is associated.

In consequence of these characteristics it can readily be understood that in order to make the profitable treatment of the ore possible, thoroughly experienced metallurgists must conduct the experiments preliminary to the commencement of mining and milling on a commercial scale. Too often this precaution has been neglected, with disastrous results.

We have, however, in the South many advantages over the West which enable us to mine and mill, profitably, ore which otherwise would have to be sent to the waste-dump. I am not ready to assert that we have anywhere mines which will compare with the Homestake or the Alaska-Treadwell in extent of the ore-bodies, and consequently, even with our low rate of wages, and low cost of mining, timbers and fuel, we cannot conduct operations at such a low expense as would compare with the operations at these mines, if such comparison be based only on actual cost. We can, however, at several locations, and especially in portions of Alabama, mine and mill for 75 cents per ton, with a crushing capacity of less than 50 tons per day. The thickness of the ore-bodies where this can be done is about 40 feet; their structure is such that they can be mined by the open-cut or quarry system to a depth of about 100 feet, and the ore is comparatively free-milling in character.

Even in localities where the natural conditions are such as to render mining more expensive we can work for from 50 to 60 per cent. less cost than is required in the West under the same conditions. This will be readily understood when I say that in these States we pay the following rates of wages:

Mine foreman,	$1.50 to $2.00	per day
Miners, underground,	1.00 to 1.25	" "
Miners, on surface and in open cuts,	.75	" "
Timbermen,	1.00 to 1.25	" "
Amalgamators,	1.50 to 2.00	" "
General mill labor, including feeders, oilers, etc.,	.75	" "
General labor,	.60 to .80	" "
Cord-wood for fuel,	.75 to 1.25	per cord
Mining timbers,	.10 to .50	per set

Silver-Losses in Cupellation.

BY L. D. GODSHALL, EVERETT, WASHINGTON.

(Colorado Meeting, September, 1896.)

A GREAT deal has been written of late regarding the loss of silver in assaying; very discordant results have been published by different writers, and much uncertainty exists concerning even approximate losses in a careful determination of silver, as represented by a fire-assay.

In the present paper it has been the aim of the writer to determine as accurately as possible the losses sustained by silver, under certain specified conditions, during the process of cupellation. Such losses are well known to be due to absorption by the cupel and volatilization. The conditions, however, which govern these losses are not believed by the writer to be so well understood. The most important of these are:

1. The effect of variable quantities of silver used.
2. The temperature of cupellation.
3. The character and quantity of impurities in the lead button.
4. The weight of the lead button.
5. The nature of the cupel.

The following experiments were made for the purpose of demonstrating, as nearly as possible, the total loss of silver in cupellation under the most favorable conditions to be obtained in commercial assaying.

Chemically pure silver and pure lead were taken. The latter was obtained from test-lead, first melted into a bar, from which pieces of the desired weight were cut and accurately weighed.

The experiments recorded in this paper were planned by the writer, but executed throughout by Mr. J. N. Walker, whose ability as an accurate, careful and painstaking assayer is too well known and recognized throughout the West to require any further endorsement from me.

The conditions governing the loss of silver in cupellation were examined as carefully and completely as was thought possible, with the exception of the comparative effect of impurities in the lead button, which will be the subject of a future paper.

A brief outline of the plan of the experiments is as follows: All assays were made in triplicate. The effect of variable quantities of silver used was determined by taking carefully weighed quantities representing approximately 2, 5, 10, 20, 50, 100 and 200 milligrammes. The effect of temperature was determined by cupelling not less than three rows in any experiment, whilst, in several, four rows were taken. Coolers for the back-row were invariably used, and the greatest possible care was taken to preserve a uniform and proper heat throughout the operation.

The effect and character of impurities in the lead button, as before stated, will not be made a part of this paper. The effect of a variable quantity of lead used was determined by taking a quarter, a half, three-quarters and a whole assay-ton for each series of experiments.

The influence of the nature of the cupel will be treated in this paper only so far as to demonstrate the reliability of the cupels used by recording the results of a series of experiments obtained by using, in addition to the cupels of our own make, two other kinds as standards. The latter were obtained from the New York Assay Office and the San Francisco Mint, through the kindness of Mr. Andrew Mason, the Superintendent of the New York Assay Office, and Mr. Ben W. Day, of the San Francisco Mint.

Three series of experiments were made to examine the effect of the variable conditions, as above stated, from every conceivable standpoint, as well as to check the results obtained.

The first series consisted in cupelling at one time not more than two different quantities of silver with a uniform weight of lead. The position of the cupels in this series was as follows:

Approximately *a* milligrammes of Silver.			Approximately *b* milligrammes of Silver.		
7	8	9	7	8	9
4	5	6	4	5	6
1	2	3	1	2	3

No. 1 representing the front row and first in each set.

In the experiments with 20 and 50 milligrammes silver four rows were cupelled at one time instead of three as shown above, otherwise the positions were the same.

In the second series the same weight of lead was again used, but seven different weights of silver were taken for each cupellation. The position of the cupels was as follows:

Milligrammes of Silver Used.

2.	5.	10.	20.	50.	100.	200.
15	16	17	18	19	20	21
8	9	10	11	12	13	14
1	2	3	4	5	6	7

No. 1 represents left hand end of front row.

In the third series not more than two different weights of silver, but four different weights of lead, were used in each cupellation. The position of the cupels was as follows:

Weight in Assay-Tons of Lead Used.

¼.	½.	¾.	1.	¼.	½.	¾.	1.
a Milligrammes of Silver.				*b* Milligrammes of Silver.			
9	10	11	12	9	10	11	12
5	6	7	8	5	6	7	8
1	2	3	4	1	2	3	4

In the final series of experiments, where the comparative absorption of the different cupels was determined, the same quantity of silver, and not more than two different weights of lead, were used in each cupellation.

The position of the cupels was as follows:

a Milligrammes of Silver Taken.

x Assay-Tons of Lead.	O_7	X_8	E_9	*y* Assay-Tons of Lead.	O_7	X_8	E_9
	X_4	E_5	O_6		X_4	E_5	O_6
	E_1	O_2	X_3		E_1	O_2	X_3

No. 1 represents first button in front row of each set.
Nos. 1, 5 and 9 represent Everett cupels (E).
Nos. 2, 6 and 7 represent N. Y. Assay Office cupels (O).
Nos. 3, 4 and 8 represent S. F. Mint cupels (X).

The results of the above series of cupellations are given in the tables appended to this paper. The buttons represented by each table were cupelled at one and the same time, excepting Tables I., II., III. and IV., in which the exceptions are the 10-mg. buttons, which were cupelled by themselves; those of Tables I. and II. in one operation, and those of Tables III. and IV. in another.

Throughout the whole of the above series, wherever it is not distinctly stated that proof-silver only was used, silver buttons were used which had been obtained from a previous cupellation, the exceptions being those marked by an asterisk in the tables.

The object in doing this was to determine if any lead had been left in the button, which would be shown by the check, containing only proof-silver, used in each case.

Throughout the foregoing experiments discrepancies in results are frequent; but it is believed by the writer that a repetition of the experiments made in the most careful manner would not show any smaller number, with one possible exception. It was discovered while conducting the above experiments that, where buttons of unequal size were cupelled at the same time, it was of the greatest importance to lower the temperature of the muffle as soon as the smaller buttons had "uncovered."

In some cases, where buttons of different sizes were all put in the muffle at the same time, and the heat was rather low for the beginning, the small buttons would "uncover" and perhaps be nearly half cupelled before the large ones started and the temperature could be lowered to the proper degree. One way

to overcome this would be to put in the large buttons first. This part of the operation is believed by the writer to be very important, as the condition is a common one where many scorification-assays are made.

In all the above experiments, the cupellations were made at a temperature as nearly uniform as possible. Dry cupels were invariably used, and were allowed to remain in the muffle until of the same temperature, before the buttons were put in.

The cupellation was started by closing the front of the muffle and allowing it to remain closed until all buttons were bright, or "uncovered," when the temperature would be lowered as quickly as possible, and the cupellation would then be conducted at a low heat until the end. The cupels, however, were not pushed back just before the buttons "brightened," nor afterwards, a condition stated as necessary in most text-books on assaying, but not found so by the writer when using pure lead and pure silver, and in quantities as above.

In some cases, where the temperature was quite low at the end, the front of the muffle would be closed just before "brightening," and left closed for several minutes, to insure the removal of all the lead.

No distinction is made in this paper between the loss by absorption and volatilization. The writer expects to continue experiments along this line, and in a future paper to discuss this matter more fully.

First Series.

Table I.—$\frac{1}{4}$ *A. T. Lead Taken for Cupellation. Proof-Silver Only Taken for this Table.*

No.	2 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	5 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	10 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	2.69	2.57	0.12	4.46	......	4.76	4.66	0.10	2.10	......	10.61	10.18	0.43	4.05	......	
2	2.76	2.61	0.15	5.43	4.69	5.81	5.65	0.16	2.76	2.97	12.22	11.75	0.47	3.85	4.08	Front Row.
3	2.64	2.53	0.11	4.17	......	5.69	5.46	0.23	4.04	......	10.58	10.12	0.46	4.35	......	
4	2.27	2.11	0.16	7.05	......	5.41	5.15	0.26	4.81	......	11.57	11.04	0.53	4.58	......	
5	2.32	2.17	0.15	6.47	8.84	5.65	5.42	0.23	4.07	4.12	11.90	11.38	0.52	4.37	4.35	Second Row.
6	2.00	1.74	0.26	13.00	......	5.76	5.56	0.20	3.47	......	11.69	11.21	0.48	4.11	......	
7	2.58	2.38	0.20	7.75	......	5.73	5.54	0.19	3.31	......	12.18	11.66	0.52	4.27	......	
8	2.48	2.37	0.11	4.44	5.74	5.26	5.06	0.20	3.80	3.71	11.80	11.52	0.28	2.37	3.15	Third Row.
9	2.79	2.65	0.14	5.02	......	5.71	5.48	0.23	4.03	......	13.58	13.20	0.38	2.80	......	
					6.42					3.60					3.86	

Table II.—$\frac{1}{2}$ *A. T. Lead Taken. All Proof-Silver for* 10 *Mgs.*

No.	2 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	5 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	10 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	*2.12	2.04	0.08	3.78	.....	*5.94	5.81	0.13	2.19	......	11.56	11.17	0.39	3.37	......	
2	2.61	2.52	0.09	3.45	3.60	5.65	5.43	0.22	3.89	2.82	12.03	11.49	0.54	4.49	3.84	Front Row.
3	2.53	2.44	0.09	3.56	......	5.46	5.33	0.13	2.38	......	10.69	10.30	0.39	3.65	......	
4	2.11	1.95	0.16	7.58	......	5.15	5.03	0.12	2.33	......	10.06	9.40	0.66	6.56	......	
5	*2.90	2.80	0.10	3.45	5.21	*5.61	5.44	0.17	3.03	3.07	11.85	11.42	0.43	3.66	4.51	Second Row.
6	1.74	1.66	0.08	4.60	.... .	5.56	5.34	0.22	3.96	......	11.80	11.41	0.39	3.31	......	
7	2.38	2.24	0.14	5.88	......	5.54	5.34	0.20	3.61	......	10.27	9.86	0.41	3.98	.....	
8	2.87	2.24	0.13	5.49	4.81	5.06	4.81	0.25	4.94	3.59	12.27	11.79	0.48	3.91	3.93	Third Row.
9	*2.28	2.21	0.07	3.07	......	*5.44	5.32	0.12	2.21	......	11.03	10.60	0.43	3.90	......	
					4.54					3.16					4.09	

Table III.—$\frac{3}{4}$ *A. T. Lead Taken.*

No.	2 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	5 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	10 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	2.04	1.89	0.15	7.35	......	5.81	5.54	0.27	4.64	......	*10.29	9.61	0.68	6.61	......	
2	*2.60	2.41	0.19	7.31	7.21	*5.50	5.23	0.27	4.91	5.00	11.75	11.05	0.70	5.96	6.03	Front Row.
3	2.44	2.27	0.17	6.97	......	5.33	5.04	0.29	5.44	......	10.12	9.56	0.56	5.53	......	
4	*2.00	1.82	0.18	9.00	......	*5.10	4.74	0.36	7.06	......	10.04	9.46	0.58	5.78	......	
5	2.80	2.62	0.18	6.43	6.95	5.44	5.16	0.28	5.14	5.63	*11.49	10.88	0.61	5.31	6.10	Second Row.
6	1.66	1.57	0.09	5.42	......	5.34	5.09	0.25	4.68	......	11.21	10.40	0.81	7.22	......	
7	2.24	2.09	0.15	6.70	......	5.34	5.10	0.24	4.49	.....	11.66	10.91	0.75	6.43	......	
8	2.24	2.07	0.17	7.59	7.08	4.81	4.62	0.19	3.95	4.57	11.52	10.72	0.80	6.94	6.52	Third Row.
9	*2.30	2.14	0.16	6.96	......	*5.50	5.21	0.29	5.27	......	*13.25	12.43	0.82	6.18	......	
					7.08					5.07					6.22	

Table IV.—1 *A. T. Lead Taken.*

No.	2 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	5 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	10 Mg. Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	1.89	1.78	0.11	5.82	......	5.54	5.36	0.18	3.25	.... .	*11.29	10.72	0.57	5.05	......	
2	*2.38	2 22	0.16	6.72	5.94	*5.16	4.98	0.18	3.49	3.50	11.49	10.78	0.71	6.18	5.17	Front Row,
3	2.27	2.15	0.12	5.28	......	5.04	4.85	0.19	3.77	......	10.30	9.86	0.44	4.27	......	
4	*1,77	1.60	0.17	9.60	......	*4.67	4.54	0.13	2.78	.....	9.40	8.96	0.44	4.68	......	
5	2.62	2.44	0.18	6.87	8.04	5.16	4.96	0.20	3.87	3.26	*11.55	10.67	0.88	7.62	6.29	Second Row.
6	1.57	1.45	0.12	7.64	......	5.09	4.93	0.16	3.14	......	11.41	10.66	0.75	6.57	......	
7	2.09	1.89	0.20	9.57	......	5.10	4.90	0.20	3.92	......	9.86	9.24	0.62	6.29	.....	
8	2.07	1.84	0.23	11,11	9.82	4.62	4.49	0.13	2.81	3.63	11.79	11.14	0.65	5.51	5.73	Third Row.
9	*2.05	1.87	0.18	8.78	......	*5.03	4.82	0.21	4.17	......	*10.77	10.19	0.58	5.38	......	
					7.93					3.46					5.73	
					6.49					3.82					4.97	

* Proof-silver.

FIRST SERIES.

TABLE V.—¼ *A. T. Lead Taken.*

No.	20 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	50 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	20.05	18.74	1.31	6.53	......	52.02	50.52	1.50	2.88	......	
2	18.17	17.28	0.89	4.90	5.68	45.01	43.80	1.21	2.69	2.12	Front Row.
3	*24 41	23.04	1.37	5.61	......	*48.84	48.45	0.39	0.80	......	
4	*17.41	16.42	0.99	5.69	......	*52.53	51.42	1.11	2.11	......	
5	20.38	19.68	0.70	3.43	4.37	49.48	48.38	1.10	2.23	2.35	Second Row.
6	23.00	22.08	0.92	4.00	......	49.63	48.28	1.35	2.72	......	
7	19.58	18.76	0.82	4.18	......	52.42	51.16	1.26	2.40	......	
8	*21.32	20.72	0 60	2.81	3.69	*48.06	46.83	1.23	2.56	2.60	Third Row.
9	19.06	18.28	0.78	4.09	......	50.50	49.06	1.44	2.85	......	
10	15.67	15.07	0.60	3.83	......	43.82	42.94	0.88	2.08	......	
11	23.49	22.75	0.74	3.15	3.33	48.26	47.38	0.88	1.82	1.81	Fourth Row.
12	*17.96	17.42	0.54	3.01	......	*53.54	52.72	0.82	1.53	......	
					4.27					2.22	

TABLE VI.—½ *A. T. Lead Taken. Proof-Silver Only in this Table.*

No.	20 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	50 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	21.74	21.08	0.66	3.03	.....	54.62	53.42	1.20	2.19	.....	
2	20.68	20.13	0.55	2.66	3.03	47.44	46.42	1.02	2.15	2.17	Front Row.
3	22.57	21.80	0.77	3.41	......	53.28	52.12	1.16	2.18	......	
4	21.08	20.05	1.03	4.89	......	52.31	50.75	1.56	2.98	......	
5	22.12	21.22	0.90	4.07	4.12	53.78	52.60	1.18	2.19	2.68	Second Row.
6	21.18	20.46	0.72	3.40	......	52.62	51.10	1.52	2.88	......	
7	22.88	20.50	2.38	10.40	......	50.50	48.80	1.70	3.38	......	
8	18.58	17.40	1.18	6.35	7.85	51.48	49.91	1.57	3.05	3.16	Third Row.
9	21.28	19.83	1.45	6.81	......	54.08	52.44	1.64	3.05	......	
10	18.70	16.45	2.25	12.03	......	49 36	45.36	4.00	8.10	......	
11	22.86	22.00	0.86	3.76	7.08	53.80	50.06	3.74	6.95	5.69	Fourth Row.
12	22.21	21.00	1.21	5.45	......	54.91	53.80	1.11	2.02	......	
					5.52					3.42	

TABLE VII.—¾ *A. T. Lead Taken. Proof-Silver Only in this Table.*

No.	20 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	50 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	22.31	20.86	1.45	6.50	......	42.88	41.86	1.02	2.38	......	
2	23.90	22.78	1.12	4.69	4.47	54.58	53.10	1.48	2.71	2.04	Front Row.
3	23.75	23.22	0.53	2.23	......	54.78	54.22	0.56	1.02	.....	
4	23 57	22.08	1.49	6.32	......	54.40	52.60	1.80	3.31	......	
5	24.58	23.56	1.02	4.15	4.70	54.41	52.80	1.61	2.96	2.80	Second Row.
6	21.48	20.70	0.78	3.63	......	54.20	53.05	1.15	2.12	......	
7	24.32	23.15	1.17	4.81	......	53.04	50.31	2.73	5.14	......	
8	22.56	21.60	0.96	4.26	4.31	53.14	51.50	1.64	3.08	3.75	Third Row.
9	22.78	21.90	0.88	3.86	......	55.20	53.52	1.68	3.04	......	
10	21.15	20.16	0.99	4.68	......	54.04	52.52	1.52	2.81	.....	
11	22.53	21.50	1.03	4.57	4.34	54.74	53.11	1.63	2.98	2.98	Fourth Row.
12	22.22	21.38	0.84	3.78	......	52.87	51.20	1.67	3.16	......	
					4.45					2.89	

TABLE VIII.—1 *A. T. Lead Taken. Proof-Silver Only in this Table.*

No.	20 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	50 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	22.30	21.34	0.96	4.30	......	50.48	48.70	1.78	3.52	......	Front Row.
2	20.28	19.67	0.61	3.01	3.50	48.41	42.06	1.35	3.11	2.94	
3	24.08	23.31	0.77	3.20	......	48.41	47.35	1.06	2.19	......	
4	24.57	23.11	1.46	5.94	......	51.37	49.38	1.99	3.87	......	Second Row.
5	25.88	24.38	1.50	5.80	5.69	48.30	46.55	1.75	3.62	3.74	
6	21.95	20.78	1.17	5.33	......	48.28	46.48	1.80	3.73	......	
7	21.88	20.80	1.08	4.94	......	51.05	49.26	1.79	3.51	......	Third Row.
8	22.72	21.50	1.22	5.37	5.46	46.79	45.25	1.54	3.29	3.32	
9	24.50	23 01	1.49	6.08	......	49.01	47.46	1.55	3.16	......	
10	21.55	20.38	1.17	5.43	......	42.92	40.97	1.95	4.54	......	Fourth Row.
11	24.46	23.30	1.16	4.74	5.14	47.35	45.26	2.09	4.41	4.16	
12	22.66	21.48	1.18	5.25	......	52.70	50.84	1.86	3.53	......	
					4.95					3.54	
					4.80					3.02	

* Proof-silver.

First Series.

Table IX.—¼ *A. T. Lead Taken. Proof-Silver Only in this Table.*

No.	100 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	200 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	108.08	106.03	2.05	1.90	......	209.72	207.04	2.68	1.28	......	
2	109.56	107.56	2.00	1.82	1.86	207.68	204.82	2.86	1.38	1.34	Front Row.
3	113.30	111.18	2.12	1.87	......	200.50	197.74	2.76	1.37	......	
4	106.57	104.36	2.21	2.07	......	210.35	207.55	2.80	1.33	......	
5	105.53	103.60	1.93	1.83	1.94	203.00	199.80	3.20	1.57	1.44	Second Row.
6	111.02	108.88	2.14	1.93	......	200.67	197.80	2.87	1.43	......	
7	108.45	106.34	2.11	1.94	......	210.20	207.78	2.42	1.15	......	
8	110.10	108.53	1.57	1.42	1.69	208.62	206.03	2.59	1.24	1.17	Third Row.
9	104.94	103.16	1.78	1.70	......	219.15	216.68	2.47	1.13	......	
					1.83					1.32	

Table X.—½ *A. T. Lead Taken.*

No.	100 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	200 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	*111.53	109.08	2.45	2.19	......	*211.34	208.00	3.34	1.58	......	
2	107.56	105.56	2.00	1.86	2.24	204.82	201.66	3.16	1.54	1.68	Front Row.
3	111.18	108.20	2.98	2.68	......	197.74	194.22	3.52	1.78	......	
4	104.36	102.90	1.46	1.40	......	207.55	203.50	4.05	1.95	......	
5	*106.63	103.91	2.72	2.55	2.00	*201.80	198.28	3.52	1.74	1.77	Second Row.
6	108.88	106.65	2.23	2.05	......	197.80	194.62	3.18	1.61	......	
7	106.34	104.10	2.24	2.11	......	207.78	204.46	3.32	1.60	......	
8	108.53	105.76	2.77	2.55	2.06	206.03	202.79	3.24	1.57	1.63	Third Row.
9	*105.66	104.04	1.62	1.53	......	*219.88	216.08	3.80	1.73	.....	
					2.10					1.68	

Table XI.—¾ *A. T. Lead Taken.*

No.	100 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	200 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	*103.58	101,62	1.96	1.89	......	*203.70	200.00	3.70	1.81	......	
2	105.56	103.34	2.22	2.10	1.95	201.66	198.12	3.54	1.75	1.75	Front Row.
3	108.20	106.20	2.00	1.85	......	194.22	190 96	3.26	1.68	......	
4	102.90	98.82	4.08	3.96	......	203.50	199.64	3.86	1.90	.....	
5	*100.88	98.65	2.23	2.21	2.85	*196.28	191.71	4.57	2.33	2.16	Second Row.
6	106.65	104.12	2.53	2.37	......	194.62	190.22	4.40	2.26	......	
7	104.10	101.12	2.98	2.86	......	204.46	200.88	4.08	1.99	......	
8	105.76	103.82	1.94	1.83	2.58	202.79	197.88	4.91	2.42	2.17	Third Row.
9	*101.54	98.45	3.09	3.04	......	*212.88	208.40	4.48	2.10	......	
					2.46					2.03	

Table XII.—1 *A. T. Lead Taken.*

No.	100 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	200 Milligrammes. Weight. Before.	After.	Silver Lost. Milli-grammes.	Per cent.	Average Per cent.	Position of Cupel.
1	*104.82	102.30	2.52	2.40	......	*202.10	198.88	3.22	1.59	......	
2	103.44	100.80	2.64	2.55	2.32	198.12	194.21	3.91	1.97	1.75	Front Row.
3	106.20	104.08	2 12	2.00	......	190.96	187.74	3.22	1.68	......	
4	98.92	95.72	3.20	3.24	......	199.64	195.20	4.44	2.22	......	
5	*102.75	100.07	2.68	2.61	3.03	*194.91	190.36	4.55	2.33	2.24	Second Row.
6	104.12	100.74	3.38	3 25	......	190.22	186.08	4.14	2.17	......	
7	101.12	97.78	3.34	3.30	......	200.38	195.67	4.71	2.35	......	
8	103.82	100.60	3.22	3.10	3.08	197.88	193 44	4.44	2.24	2.34	Third Row.
9	*100.65	97.86	2.79	2.77	......	*213.58	208.36	5.22	2.44	......	
					2.81					2.11	

* Proof-silver.

SECOND SERIES.

TABLE I.—$\frac{1}{4}$ *A. T. Lead Taken.*

No.	2 Milligrammes.					5 Milligrammes.					10 Milligrammes.					20 Milligrammes.					50 Milligrammes.					100 Milligrammes.					200 Milligrammes.				
	Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.		
	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.
1	*2.60	2.53	0.07	2.69	......	5.80	5.59	0.21	3.62	......	11.26	10.84	0.42	3.73	......	25.02	24.61	0.41	1.64	......	*51.50	50.45	1.05	2.04	......	100.64	98.38	2.26	2.24	......	196.72	192.88	3.84	1.95	......
2	3.32	3.20	0.12	3.61	3.73	*5.59	5.35	0.24	4.29	3.72	11.29	10.91	0.38	3.36	3.01	*24.00	23.50	0.50	2.08	1.86	*50.48	49.38	1.10	2.17	2.00	*106.48	104.69	1.79	1.68	1.87	200.42	197.90	2.52	1.25	1.46
3	3.47	3.30	0.17	4.90	......	5.53	5.35	0.18	3.26	......	*10.88	10.67	0.21	1.93	......	21.61	21.21	0.40	1.85	......	51.50	50.58	0.92	1.78	......	105.66	103.88	1.78	1.68	......	*204.80	201.86	2.44	1.19	......

TABLE II.—$\frac{1}{2}$ *A. T. Lead Taken. Fine Silver only Taken for this Table.*

No.	2 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	5 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	10 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	20 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	50 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	100 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	200 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.
1	2.66	2.48	0.18	6.77	......	5.98	5.80	0.18	3.01	......	11.54	11.26	0.28	2.43	......	25.60	25.02	0.58	2.26	......	52.80	51.38	1.42	2.69	......	102.42	100.64	1.78	1.74	......	200.40	196.72	3.68	1.83	......
2	3.54	3.32	0.22	6.22	6.90	5.34	5.39	0.25	4.43	4.46	11.62	11.29	0.33	2.84	3.44	24.62	23.70	0.92	3.73	2.82	51.82	50.48	1.34	2.59	2.67	108.68	106.37	2.31	2.12	2.03	204.18	200.42	3.76	1.84	1.73
3	3.76	3.47	0.29	7.71	......	5.88	5.53	0.35	5.95	......	11.30	10.73	0.57	5.04	......	22.16	21.61	0.55	2.48	......	52.94	51.50	1.44	2.72	......	108.08	105.66	2.42	2.24	......	206.43	203.26	3.17	1.53	......

TABLE III.—$\frac{3}{4}$ *A. T. Lead Taken.*

No.	2 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	5 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	10 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	20 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	50 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	100 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	200 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.
1	*2.41	2.26	0.15	6.22	......	5.59	5.38	0.21	3.76	......	10.84	10.45	0.39	3.60	......	24.61	23.67	0.94	3.82	......	*50.33	49.31	1.02	2.03	......	98.38	96.66	1.72	1.75	......	192.88	190.04	2.84	1.47	......
2	3.20	2.90	0.30	9.37	8.43	*5.15	4.88	0.27	5.24	4.68	10.91	10.23	0.68	6.23	5.37	*23.20	22.23	0.97	4.18	4.41	49.38	47.62	1.76	3.56	3.02	*104.58	101.63	2.95	2.82	2.40	197.50	193.52	3.98	2.01	1.85
3	3.30	2.98	0.32	9.70	......	5.35	5.08	0.27	5.05	......	*10.52	9.86	0.66	6.27	......	21.21	20.10	1.11	5.23	......	50.58	48.82	1.76	3.48	......	103.88	101.14	2.74	2.64	......	*200.82	196.63	4.19	2.08	......

TABLE IV—1 *A. T. Lead Taken.*

No.	2 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	5 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	10 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	20 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	50 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	100 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	200 mg Before.	After.	Milli-grammes.	Per cent.	Average Per cent.
1	*2.37	2.21	0.16	6.75	......	5.38	5.11	0.27	5.02	......	10.45	10.02	0.43	4.11	......	23.67	22.89	0.78	3.30	......	*49.35	48.06	1.29	2.61	......	96.66	94.65	2.01	2.08	......	190.04	185.46	4.58	2.41	......
2	2.90	2.61	0.29	10.00	9.39	*4.98	4.60	0.38	7.63	7.29	10.23	9.56	0.67	6.55	5.44	*22.00	20.84	1.16	5.27	4.51	47.62	45.85	1.77	3.72	3.27	*101.83	98.70	3.13	3.07	2.64	193.52	189.36	4.16	2.15	2.24
3	2.98	2.64	0.34	11.41	......	5.08	4.61	0.47	9.23	......	*9.90	9.34	0.56	5.65	......	20.10	19.10	1.00	4.97	......	48.82	47.12	1.70	3.48	......	101.14	98.35	2.79	2.76	......	*196.70	192.46	4.24	2.16	......
					7.11					5.04					4.31					3.40					2.74					2.23					1.82

* Proof-silver.

Third Series.

Table I.—2 *Milligrammes Silver Taken.* *Proof-Silver Taken for Whole Series.*

No.	¼ A. T. Pb. Weight. Before.	¼ A. T. Pb. Weight. After.	¼ A. T. Pb. Silver Lost. Milli-grammes.	¼ A. T. Pb. Silver Lost. Per cent.	¼ A. T. Pb. Silver Lost. Average Per cent.	½ A. T. Pb. Weight. Before.	½ A. T. Pb. Weight. After.	½ A. T. Pb. Silver Lost. Milli-grammes.	½ A. T. Pb. Silver Lost. Per cent.	½ A. T. Pb. Silver Lost. Average Per cent.	¾ A. T. Pb. Weight. Before.	¾ A. T. Pb. Weight. After.	¾ A. T. Pb. Silver Lost. Milli-grammes.	¾ A. T. Pb. Silver Lost. Per cent.	¾ A. T. Pb. Silver Lost. Average Per cent.	1 A. T. Pb. Weight. Before.	1 A. T. Pb. Weight. After.	1 A. T. Pb. Silver Lost. Milli-grammes.	1 A. T. Pb. Silver Lost. Per cent.	1 A. T. Pb. Silver Lost. Average Per cent.	Average of All.
1	2.78	2.68	0.10	3.60	......	2.47	2.40	0.07	2.82	......	3.20	3.10	0.10	3.12	......	2.89	2.74	0.15	5.19	......	
2	2.78	2.65	0.13	4.68	4.05	2.94	2.84	0.10	3.40	4.33	2.56	2.49	0.07	2.73	4.51	3.26	3.10	0.16	4.91	6.49	4.85
3	2.39	2.31	0.08	3.35	......	3.07	2.86	0.21	6.84	......	2.96	2.79	0.17	5.74	......	2.82	2.64	0.18	6.38	......	
4	2.62	2.50	0.12	4.58	......	2.80	2.68	0.12	4.28	......	2.48	2.32	0.16	6.45	......	2.00	1.81	0.19	9.50	......	

Table II.—5 *Milligrammes Silver Taken.*

1	5.48	5.30	0.18	3.28	......	5.32	5.21	0.11	2.07	. ...	4.78	4.65	0.13	2.72	......	5.00	4.78	0.22	4.40	.....	
2	5.32	5.10	0.22	4.14	3.52	5.48	5.26	0.22	4.01	3.15	5.96	5.69	0.27	4.53	4.25	5.22	4.99	0.23	4.41	4 84	3.94
3	5.24	5.08	0.16	3.05	......	5.98	5.81	0.17	2.84	......	5.06	4.77	0.29	5.78	......	6.04	5.64	0.40	6.62	......	
4	5.84	5.63	0.21	3.60	......	5.73	5.52	0.21	3.66	......	5.46	5.24	0.22	4.03	......	5.61	5.39	0.22	3.92	......	

Table III.—10 *Milligrammes Silver Taken.*

1	10.36	10.01	0.35	3.38	......	10.84	10.58	0.26	2.39	......	10.54	10.14	0.40	3.79	.....	11.46	11.06	0.40	3.49	......	
2	12.86	12.25	0.61	4.74	4.01	12.10	11.59	0.51	4.21	3.19	13.04	12.50	0.54	4.14	3.87	12.00	11.40	0.60	5.00	4.15	3.79
3	11.55	11.10	0.45	3.90	......	11.84	11.49	0.35	2.96	......	11.40	10.98	0.42	3.68	......	11.59	11.13	0.46	3.97	......	

Table IV.—20 *Milligrammes Silver Taken.*

1	21.12	20.50	0.62	2.93	......	21.25	20.88	0.37	1.74	......	21.76	21.22	0.54	2.48	......	20.78	20.26	0.52	2.50	......	
2	23.63	22.74	0.89	3.77	3.26	21.81	21.10	0.71	3.26	2.80	22.52	21.69	0.83	3.68	2.88	22.06	21.07	0.99	4.49	3.33	3.07
3	22.09	21.41	0.68	3.08	......	21.50	20.77	0.73	3.40	......	22.70	22.14	0.56	2.47	......	22.55	21.77	0.78	3.01	......	

Table V.—50 *Milligrammes Silver Taken.*

1	50.06	48.92	1.14	2.27	......	55.42	54.02	1.40	2.53	......	51.66	50.16	1.50	2.98	......	55.16	53.44	1.72	3.12	......	
2	50.11	48.71	1.40	2.79	2.44	55.66	54.23	1.43	2.39	2.71	54.75	52.70	2.05	3.74	3.17	51.88	49.60	2.28	4.39	3.50	2.96
3	53.72	52.50	1.22	2.27	......	51.26	49.61	1.65	3.22	......	54.78	53.23	1.55	2.83	.. ...	52.33	50.76	1.57	3.00	......	

Table VI.—100 *Milligrammes Silver Taken.*

1	114.20	111.65	2.55	2.23	......	101.88	99.69	2.19	2.15	......	107.28	104.50	2.78	2.59	......	101.72	99.24	2.48	2.44	......	
2	108.59	106.29	2.30	2.11	2.17	102.49	100.23	2.26	2.20	2.85	110.02	106.76	3.26	2.96	2.69	101.66	97.84	3.82	3.75	2.95	2.54
3	102.10	99.88	2.22	2.17	......	100.40	97.67	2.73	2.71	.....	105.34	102.70	2.64	2.51	......	107.34	104.48	2.86	2.66	......	

Table VII.—200 *Milligrammes Silver Taken.*

1	201.30	197.58	3.72	1.85	......	218.18	215.01	3.17	1.45	......	210.92	207.48	3.44	1.68	......	214.93	210.49	4.44	2.06	......	
2	207.40	203.86	3.54	1.71	1.76	207.00	202.76	4.24	2.05	1.91	203.68	199.33	4.35	2.18	1.87	204.32	199.83	4.49	2.19	1.96	1.87
3	214.88	211.21	3.67	1.71	......	203.19	198.68	4.51	2.22	......	205.76	201.95	3.81	1.85	......	204.76	201.42	3.34	1.63	......	

Summary of Average Results Obtained in First, Second and Third Series, Expressed in Per Cent.

Position of Cupel.	Approximate Amount of Silver in Milligrammes and Exact Amount of Lead in Assay-Tons.																																		
	2 Milligrammes.					5 Milligrammes.					10 Milligrammes.					20 Milligrammes.					50 Milligrammes.					100 Milligrammes.					200 Milligrammes				
	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.
First Series.																																			
Front......	4.69	3.60	7.21	5.94	5.36	2.97	2.82	5.00	3.50	3.57	4.08	3.84	6.03	5.17	4.78	5.68	3.03	4.47	3.50	4.17	2.12	2.17	2.04	2.94	2.32	1.86	2.24	1.95	2.32	2.09	1.34	1.63	1.75	1.75	1.62
Second ...	8.84	5.21	6.95	8.04	7.26	4.12	3.07	5.63	3.26	4.02	4.35	4.51	6.10	6.29	5.31	4.87	4.12	4.70	5.69	4.72	2.35	2.68	2.80	3.74	2.89	1.94	2.00	2.85	3.03	2.46	1.44	1.77	2 16	2.24	1.90
Third......	5.74	4.81	7.08	9.82	6.86	3.71	3.59	4.57	3.63	3.88	3.15	3.93	6.52	5.73	4.83	3.69	7.85	4.31	5.46	5.33	2.60	3.16	3.75	3.32	3.21	1.69	2.06	2.58	3.08	2.35	1.17	1.63	2.17	2.34	1.83
Fourth....	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	3.33	7.08	4.34	5.14	4.97	1.81	5.69	2.98	4.16	3.66	......	......	......	......	......	......	......	......	......	......
Average..	6.42	4.54	7.08	7.93	6.49	3.60	3.16	5.07	3.46	3.82	3.86	4.09	6.22	5.73	4.97	4.27	5.52	4.45	4.95	4.80	2.22	3 42	2.89	3.54	3.02	1.83	2.10	2.46	2.81	2.30	1.32	1.68	2.03	2.11	1.78
Second Series.																																			
Front......	2.69	6.77	6.22	6.75	5.61	3.62	3.01	3.76	5.02	3.85	3.73	2.48	3.60	4.11	3.47	1.64	2.26	3.82	3.30	2.75	2.04	2.69	2.03	2.61	2.34	2.24	1.74	1.75	2.08	1.95	1.95	1.83	1.47	2.41	1.92
Second....	3.61	6.22	9.37	10.00	7.30	4.29	4.43	5.24	7.63	5.40	3 86	2.84	6.23	6.55	4.74	2.08	3.73	4.18	5.27	3.82	2.17	2.59	3.56	3.72	3.01	1.68	2.12	2.82	3.07	2.42	1.25	1.84	2.01	2.15	1.81
Third......	4.90	7.71	9.70	11.41	8.43	3.26	5.95	5.05	9.23	5.87	1.93	5.04	6.27	5.65	4.72	1.85	2.48	5.23	4.97	3.63	1.78	2.72	3.48	3.48	2.87	1.68	2.24	2.64	2.76	2.33	1.19	1.53	2.08	2.16	1.74
Average..	3.73	6.90	8.43	9.39	7.11	3.72	4.46	4.68	7.29	5.04	3.01	3.44	5.37	5.44	4.31	1.86	2.82	4.41	4.51	3.40	2.00	2.67	3.02	3.27	2.74	1.87	2.03	2.40	2.64	2.23	1.46	1.73	1.85	2.24	1.82
Third Series.																																			
Front......	3.60	2.82	3.12	5.19	3.68	3.28	2.07	2.72	4.40	3.12	3.38	2.89	3.79	3.49	3.26	2.93	1.74	2.48	2.50	2.41	2.27	2.53	2.93	3.12	2.71	2.23	2.15	2.59	2.44	2.35	1.85	1.45	1.63	2.06	1.75
Second ...	4.68	3.40	2.73	4.91	3.93	4.14	4.01	4.53	4.41	4.27	4.74	4.21	4.14	5.00	4.52	3.77	3.26	3.65	4.49	3.80	2.79	2.89	3.74	4.39	3.33	2.11	2.20	2.96	3.75	2.75	1.71	2.05	2.13	2.19	2.02
Third......	3.35	6.84	5.74	6.38	5.58	3.05	2.84	5.73	6.62	4.56	3.90	2.96	3.68	3.97	3.63	3.08	3.40	2.47	3.01	2.99	2.27	3.22	2.83	3.00	2.83	2.17	2.71	2.51	2.66	2.51	1.71	2.22	1.85	1.63	1.85
Fourth....	4.58	4.28	6.45	9.50	6.20	3.60	3.66	4.03	3.92	3.80	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
Average..	4.05	4.33	4.51	6.49	4.85	3.52	3.15	4.25	4.84	3.94	4.01	3.19	3.87	4.15	3.80	3.26	2.80	2.88	3.33	3.07	2.44	2.71	3.17	3.50	2.96	2.17	2.35	2.69	2.95	2.54	1.76	1.91	1.87	1.96	1.87
Summary of Final Average Results Obtained in the Above Three Series Expressed in Per Cents.																																			
Front......	3.66	4.40	5.52	5.96	4.89	3.29	2.63	3.83	4.31	3.51	3.73	2.89	4.47	4.26	3.84	3.42	2.34	3.59	3.10	3.09	2.14	2.46	2.33	2.89	2.45	2.11	2.04	2.10	2.28	2.13	1.71	1.64	1.62	2.07	1.76
Second ...	5.71	3.94	6.35	7.65	5.91	4.18	3.84	5.13	5.10	4.56	4.15	3.85	5.49	5.95	4.86	3.41	3.70	4.19	5.15	4.11	2.44	2.55	3 37	3.95	3.08	1.91	2.10	2.88	3.28	2.54	1.47	1.89	2.10	2.19	1.91
Third......	4.66	6.45	7.51	9.20	6.95	3.34	4.13	5.12	6.49	4.77	2.99	3.98	5.49	5.12	4.39	2.87	4.58	4.00	4.48	3.98	2.22	3.03	3.35	3.27	2.97	1.85	2.34	2.58	2.83	2.40	1.36	1.79	2.03	2.04	1.80
Average..	4.68	4.93	6.46	7.60	5.92	3.60	3.53	4.69	5.30	4.28	3.62	3.57	5.15	5.11	4.36	3.23	3.54	3.93	4.24	3.73	2.27	2.68	3.02	3.37	2.83	1.96	2.16	2.52	2.80	2.36	1.51	1.77	1.92	2.10	1.82

TABLE A.—*Experiment to Determine the Comparative Absorption of Cupels, Using ¼ A. T. Lead and Variable Quantities of Fine Silver.*

No.	Cupel Used.	2 Milligrammes Silver.					5 Milligrammes Silver.					10 Milligrammes Silver.					50 Milligrammes Silver.					200 Milligrammes Silver.				
		Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.			Weight.		Silver Lost.		
		Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	Before.	After.	Milli-grammes.	Per cent.	Average Per cent.	After.	Before.	Milli-grammes.	Per cent.	Average Per cent.
1	Everett.	2.86	2.72	0.14	4.89	......	5.53	5.37	0.16	2.89	......	16.73	16.23	0.50	2.99	......	56.22	54.86	1.36	2.42	......	201.64	197.96	3.68	1.82	......
5	"	2.40	2.29	0.11	4.58	4.57	5.88	5.68	0.20	3.40	3.33	12.26	11.94	0.32	2.61	2.62	54.42	53.15	1.27	2.33	2.40	214.04	210.59	3.45	1.61	1.59
9	"	2.36	2.26	0.10	4.24	......	5.67	5.46	0.21	3.70	......	12.80	12.51	0.29	2.27	......	52.89	51.60	1.29	2.44	......	206.04	203.30	2.74	1.33	......
2	Assay	2.85	2.77	0.08	2.81	......	5.78	5.54	0.24	4.15	......	13.11	12.66	0.45	3.43	......	51.69	50.45	1.24	2.40	......	209.58	206.18	3.40	1.62	......
6	Office,	2.44	2.33	0.11	4.51	4.19	5.41	5.21	0.20	3.70	4.32	12.89	12.50	0.39	3.03	3.44	53.92	52.56	1.36	2.52	2.61	210.91	206.87	4.04	1.91	1.74
7	New York.	2.11	2.00	0.11	5.26	......	5.28	5.01	0.27	5.11	......	12.71	12.22	0.49	3.86	......	54.48	52.90	1.58	2.90	......	210.08	206.50	3.58	1.70	......
3	Mint,	2.70	2.60	0.10	3.70	......	5.52	5.38	0.14	2.54	......	11.77	11.50	0.27	2.29	......	52.16	50.58	1.58	3.03	......	209.26	206.43	2.83	1.35	......
4	San Fran-	2.84	2.71	0.13	4.58	4.03	5.66	5.42	0.24	4.24	3.63	11.30	10.94	0.36	3.19	2.40	53.90	52.52	1.38	2.56	2.68	204.78	201.16	3.62	1.77	1.56
8	cisco.	2.62	2.52	0.10	3.82	......	5.59	5.36	0.23	4.11	......	12.80	12.58	0.22	1.72	......	53.92	52.60	1.32	2.45	......	201.28	198.14	3.14	1.56	......
						4.25					3.76					2.82					2.56					1.63

TABLE B.—*Same as Above, Except Using ½ A. T. Lead.*

1	Everett.	2.72	2.60	0.12	4.41	......	5.61	5.44	0.17	3.03	......	12.14	11.78	0.36	2.96	......	53.24	51.88	1.36	2.53	......	206.98	203.00	3.98	1.92	......
5	"	2.70	2.54	0.16	5.92	5.12	5.56	5.30	0.26	4.68	4.02	12.74	12.43	0.31	2.43	2.46	54.83	53.27	1.56	2.84	2.46	212.50	208.49	4.01	1.89	1.81
9	"	2.78	2.64	0.14	5.04	......	5.72	5.47	0.25	4.37	......	11.01	10.79	0.22	2.00	......	55.44	54.32	1.12	2.02	......	213.65	210.20	3.45	1.61	......
2	Assay	2.46	2.34	0.12	4.88	......	5.68	5.49	0.19	3.35	......	11.89	11.52	0.37	3.11	......	53.50	52.38	1.12	2.09	.....	202.70	199.00	3.70	1.82	......
6	Office,	2.88	2.75	0.13	4.51	4.86	5.53	5.28	0.25	4.52	3.99	12.97	12.70	0.27	2.08	2.79	51.04	49.74	1.30	2.55	2.27	207.42	203.00	4.42	2.13	1.95
7	New York.	2.12	2.01	0.11	5.19	......	5.59	5.36	0.23	4.11	......	12.30	11.91	0.39	3.17	......	53.69	52.52	1.17	2.18	......	210.79	206.76	4.03	1.91	......
3	Mint,	2.52	2.44	0.08	3.17	......	5.44	5.43	0.01(?)	0.18(?)	......	11.46	11.32	0.14	1.22	......	54.93	53.88	1.05	1.91	......	210.28	206.60	3.68	1.75	......
4	San Fran-	2.54	2.40	0.14	5.51	4.81	5.00	4.72	0.28	5.60	3.47	11.66	11.34	0.32	2.74	2.08	51.84	50.53	1.31	2.53	2.26	210.23	206.34	3.89	1.85	1.76
8	cisco.	2.57	2.42	0.15	5.83	......	5.60	5.34	0.26	4.64	......	10.57	10.33	0.24	2.27	......	53.46	52.21	1.25	2.34	......	205.04	201.58	3.46	1.69	......
						4.94					3.83					2.44					2.33					1.84

The Occurrence and Behavior of Tellurium in Gold-Ores, More Particularly with Reference to the Potsdam Ores of the Black Hills, South Dakota.

BY FRANK CLEMES SMITH, RAPID CITY, S. D.

(Colorado Meeting, September, 1896.)

THE study of the so-called refractory gold-ores of the Potsdam sandstone, ores which are probably of wider occurrence and of much greater economic importance in the Black Hills than is generally supposed, has led the writer to formulate in the present paper some of the results obtained in the course of a special examination of these particular ores, conducted at the State School of Mines at Rapid City, So. Dak. Various circumstances have interrupted the work, and it will easily be seen that relatively little has yet been accomplished; but it is hoped that this little may be of sufficient interest to justify a later supplement.

Tellurium, discovered by Müller von Reichenstein in 1781, is generally classed as a very rare metal; it is further mentioned by one of our late writers as "of comparatively little importance," and as obtained from its ores by a "complicated process." Concerning its rarity, we have some trustworthy evidence below. Its commercial importance, particularly in connection with electrical appliances, has been passed upon by Mr. Edison;* and it is certain that, in some metallurgical processes for the treatment of gold- and silver-ores, it may be saved as a by-product at a relatively reasonable expense.

The importance of tellurium as a mineralizing agent is emphasized by the remarks of one of our leading metallurgists,† who, in his own practice and by a wide series of experiments, has found its occurrence to be much more general than has been supposed. Recently the same writer has discussed the occurrence of tellurides and oxidized tellurium compounds in the gold-ores of Cripple Creek, Colo., and in those of the Black

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* *Trans.*, xviii., 442.

† Dr. Richard Pearce, *Trans.*, xviii., 449.

Hills.* Several important gold-producing districts may be cited in which the precious metals are known to be largely associated with tellurium.

The presence of tellurium in such ores has frequently been a source of perplexity to metallurgists, particularly in American practice. In some cases they have detected it; but, doubtless, in the majority of cases, they have not done so, or, if they have recognized it, have ignored it, with disastrous consequences. It is not, however, suprising that the metallurgy of tellurium compounds has been little studied in this country, since, until comparatively recent times, few large deposits of such ores have been worked to any great extent, and the ores of these would be of such grade as would naturally be sent to a smelter, there to be lost in the mass of others. It has been proved that telluriferous ores are not susceptible to amalgamation or to the ordinary methods of hydraulic concentration. They cannot be smelted by themselves without an unusual proportion of loss. In various details of the chlorination and cyanide processes, as practiced to-day, there is also a series of losses from volatilization of the precious metals, and these losses are heavily increased by absorption of the precious metals by melting vehicles and entrainment in waste products.

It is stated by Küstel† that "not all tellurium combinations with gold lose gold to a notable extent while roasting; but some do, and that up to a considerable amount, 20 per cent., perhaps even more." He further says, "The loss is no mechanical one, occasioned by the draft of the furnace, but principally by volatilization." Lock‡ states that in the roasting of a smelting-mixture containing small quantities of cobalt, nickel, antimony, arsenic, tellurium, gold and silver, portions of the tellurium and gold are volatilized. He cites the result of a roasting of 14.5 pounds of tellurium-ore in a muffle-furnace for 1½ hours, with the loss of 0.35 per cent. of the gold, and 3.8 per cent. of the silver; the further treatment of the roasted product by chlorination, precipitation of the gold with ferrous sulphate, precipitation of the tellurium with metallic zinc, and "examination of the argentiferous residues," gave certain results,

* *Proc. Colo. Sci. Soc.*

† *Roasting of Gold- and Silver-Ores*, p. 57.

‡ *Gold*, p. 1140.

showing only that there were serious errors in the assays (doubtless due to the presence of the very metal whose effect was under examination), somewhat reflecting upon the value of the whole investigation. Becker cites the same test.*

The resultant of various investigations, from those of Plattner up to the latest, seems to be that, in the ordinary oxidizing-roasting of gold-ores, where no tellurium is present, there is no loss of gold other than a mechanical one. On the other hand, in considering the treatment of gold-ores carrying an appreciable amount of tellurium, in any form, it seems safe to say that in any operation involving a roasting of such ores, or of products from which either the gold or the tellurium has not been separated, there will be an appreciable loss of gold by volatilization. This loss is easily noticeable in the fire-assay of such ores; and a consideration of the route taken by the gold in the course of the various metallurgical operations will show that in many cases the loss may be progressive, and not entirely due to one portion of the process. Careful experiment with such ores, especially when conducted at reduction-works handling large quantities of them, should not only finally avoid embarrassment from the presence of the tellurium, but even turn it to profit.

The discovery of tellurium in the Potsdam ores of the Black Hills was first made in the course of an examination of some samples from the Welcome mine, belonging to the Horse Shoe Mining Co. With the idea that this metal, if of general occurrence in the Potsdam ores, would go far to explain the source of their mineralization, and that it must also exert a very notable effect upon their metallurgy, a large number of samples, from various locations in the hills, were examined, and, of all these ores, including several mineralized dike-rocks from the vicinity of Potsdam ore-bodies, but one sample (later referred to in consideration of Fig. 2) failed to show the presence of tellurium.

Assays of Tellurium-Ores.

As a preliminary step toward the investigation of the ores in question, it was determined to examine the effect of various conditions upon the fire-assay, with the view of establishing a safe crucible charge. Later writers upon assaying prefer the

* *L'Or*, p. 270 *et seq.*

scorification-assay for tellurium-ores, using generally one-tenth of an assay-ton of ore to a charge. With very rich, unoxidized gold-ores of the Potsdam sandstone, in the Black Hills, the crucible-assay seems preferable, both by reason of the comparative results obtained by the two methods, and also because it is desirable to work upon a greater amount of ore than one-tenth of an assay-ton, especially in cases where the ores *may* be of low grade.

Two samples of Potsdam ore were selected, which were supposed to be of fairly high grade. One was a so-called "blue" (unoxidixed) ore and the other was a "red" (oxidized) ore. These were dried, crushed to 60-mesh and assayed by scorification- and crucible-test, using the charges ordinarily used for such ores, as follows:

Charge No. 1.

Crucible.	A. T.	Scorification.	A. T.
Ore,	0.50	Ore,	0.25
Soda bicarbonate,	1.50	Granulated lead, mixed,	2.00
Litharge,	2.00	Granulated lead on top,	1.00
Argol,	0.05	Pinch of borax.	
Borax cover.			

Results.

	Crucible. Ozs. per ton. Gold.	Crucible. Ozs. per ton. Silver.	Scorification. Ozs. per ton. Gold.	Scorification. Ozs. per ton. Silver.
Red ore,	20.34	2.66	16.20	1.70
Blue ore,	2.18	3.64	2.08	3.44

The same ores were then assayed by both methods, using the following charges:

Charge No. 2.

Crucible.	A. T.	Scorification.	A. T.
Ore,	0.25	Ore,	0.25
Soda bicarbonate,	1.00	Granulated lead, mixed,	2.00
Litharge,	2.00	Granulated lead on top,	1.00
Argol,	0.05	Litharge cover.	
Borax cover.		Pinch of borax.	

Results.

	Crucible. Ozs. per ton. Gold.	Crucible. Ozs. per ton. Silver.	Scorification. Ozs. per ton. Gold.	Scorification. Ozs. per ton. Silver.
Red ore,	22.16	3.04	21.40	2.60
Blue ore,	2.24	4.12	2.15	4.08

Both samples were then tested by crucible-assay alone, as follows:

Charge No. 3.

	Crucible. A. T.
Ore,	0.25
Soda bicarbonate,	1.00
Litharge,	2.00
Argol,	0.10
Borax cover.	

Results.

	Ozs. per ton. Gold.	Silver.
Red ore,	22.81	3.16
Blue ore,	2.31	4.13

Further alterations in the charge, consisting in increase of litharge or of argol, failed to give increased results, and charge No. 3 was deemed satisfactory.

Another sample of red ore was assayed by crucible with this charge, and also by crucible and by scorification with Furman's charges, as follows:

Furman's Charges.

	Crucible.		Scorification.
	A. T.		A. T.
Ore,	0.5	Ore,	0.10
	Grammes.		Grammes.
Soda bicarbonate,	20	Granulated lead,	50.0
Litharge,	80	Borax glass,	0.3
Granulated lead,	50	Litharge cover.	
Salt cover.			

Results.

	Ozs. per ton. Gold.	Silver.
Crucible-charges, No. 3,	4.3	4.8
Crucible-charge, Furman's,	4.06	4.59
Scorification charge, Furman's,	4.05	4.55

A sample of high-grade Cripple Creek telluride-ore was next prepared and assayed with the use of (1) the ordinary crucible-charge; (2) crucible-charge No. 3; (3) the ordinary scorification-charge; and (4) an improved scorification-charge; four assays being made by each method, under similar conditions as to temperature and time, and four beads from each charge being weighed together, re-cupelled into one with addition of silver, and parted.

Charges.

	Ordinary Crucible Charge. A. T.	Improved Scorification Charge. A. T.	Ordinary Scorification Charge. A. T.
Ore, ...	0.25	0.1	0.1
Litharge, . . .	1.00	1 (cover).	
Soda bicarbonate, .	1.00		
Argol, ...	0.05		
Granulated lead, .		2	1.5
Borax glass . .	cover.	cover.	cover.

Results.

	Ozs. per ton. Gold.	Silver.
Ordinary crucible charge, ...	52.8	3.975
No. 3, ...	54.905	4.945
Ordinary scorification charge, ...	48.75	2.9
Improved scorification charge, ...	55.82	4.608

Attempts to improve the yield of crucible-charge No. 3 by various alterations gave negative results.

It will be noticed that the results vary widely, and that the losses shown in the use of the ordinary charges are so great as to render this matter of the assay of these ores exceedingly important. In general, if the assayer obtains a good melt with clean, glassy slag and a clean, malleable button, he is apt to be content with his result; all of the above assays gave good clean slag and clean malleable buttons, yet the loss in gold varied to a maximum of about 29 per cent.

Although it would have been interesting, no attempt was made to recover the gold and silver absorbed by the cupels, and thus to determine by difference the amount lost by volatilization; since, as before stated, these experiments were made to determine an assay-charge, if possible a *crucible*-charge, which could be safely used upon a sufficient quantity of ore to give weighable returns with low-grade ores. It is probable that the percentages of loss by absorption and volatilization would have been relatively similar to those determined later on.

It was considered as a result of the above experiments that crucible-charge No. 3 could be safely used upon the Potsdam gold-ores, at least upon such of them as were under examination, although for rich unoxidized tellurides the improved scorification-charge (practically the same as Furman's) is undoubtedly preferable, making use of a smaller amount of ore.

Examination of Potsdam Ores.—Two samples of ordinary Potsdam ores were taken, one a red ore apparently completely

oxidized, the other a blue ore showing pyrite, fluorite and gypsum. Microscopic sections were made, magnified about 10 diameters and photographed. (See Figs. 1 and 2.) Under the microscope, No. 1 (Fig. 1) showed a thoroughly reorganized sandstone containing some limonite, probably from the oxidation of pyrite, and innumerable veinlets of quartz which present a beautiful comb-structure.

No. 2 (Fig. 2) presents under the microscope the appearance of a sandstone highly altered by silicification. The section shows many druses, lined with innumerable small quartz crystals, and containing calcite, fluorite and pyrite. Much pyrite is elsewhere visible.

Assays of these ores before and after screening gave the following results:

	Mesh.	Per cent. of weight.	Ozs. per ton. Gold.	Ozs. per ton. Silver.
Red ore,—No. 1, . .	not screened	100.0	0.574	2.875
	coarser than 20	0.8	0.562	2.920
	20 to 40	19.2	0.581	2.845
	40 to 60	11.2	0.556	2.890
	60 to 80	16.8	0.560	2.875
	80 to 100	7.2	0.565	2.830
	finer than 100	44.8	0.582	2.760
Blue ore,—No. 2, . .	not screened	100.00	0.325	1.055
	coarser than 20	0.23	0.330	1.02
	20 to 40	34.47	0.315	1.04
	40 to 60	17.96	0.295	1.00
	60 to 80	12.50	0.320	1.03
	80 to 100	10.93	0.320	1.02
	finer than 100	23.91	0.330	1.05

The results of the above assays indicate a very even distribution of the precious metals throughout the ore. Analyses of the ore give the following results:

Red Ore,—No. 1.

		Per cent.
SiO_2,		84.45
Al_2O_3,		4.07
Fe_2O_3,		7.28
CaO,		0.85
MgO,		0.25
SO_3,		3.71
Te,	8.426 ozs. per ton	
Au,	0.574 " " "	
Ag,	2.875 " " "	
		100.61

Blue Ore,—No. 2.

		Per cent.
SiO_2,		68.748
Al_2O_3,		3.072
Fe,		13.289
S,		11.728
$CaSO_4$, $2H_2O$ (gypsum),		0.833
CaF_2,		0.784
P_2O_5,		0.842
Te,	4.03 ozs. per ton	
Au,	0.325 " " "	
Ag,	1.055 " " "	
		99.296

An examination of the above analyses, which may be considered as typical, indicates that oxidizing action, in changing a blue to a red ore, has only found it necessary to form soluble salts chiefly of the pyrite, thereby removing iron and sulphur and increasing the per cent. of silica.

Experiments in Roasting.

Samples were taken of a very rich telluride-ore from Boulder County, Colo., and of a rich red Potsdam ore; the Colorado ore showing much metallic substance and the Potsdam ore showing none. Of each of these samples two one-fourth assay-ton charges were taken; one charge of each ore was roasted in the muffle for fifteen minutes at a cherry-red; the four charges were then made up according to crucible-charge No. 3, smelted and cupelled at the same time, with the following assay-results:

	Ozs. per ton. Gold.	Silver.
Raw telluride-ore,	180.66	2041.82
Roasted "	166.6	2022.36
Loss by roasting,	14.06	29.46
Raw Potsdam ore,	22.80	3.16
Roasted "	22.76	2.84
Loss by roasting,	0.04	0.32

In roasting the Colorado ore puffed up greatly. The slag from the raw sample was red and transparent, while the slag from the roasted sample was dove-colored and opaque.

A rich Cripple Creek telluride-ore was then taken. Two one-fourth assay-ton charges were smelted with crucible-charge No. 3, one sample having been previously roasted at a cherry-red for fifteen minutes. The lead button from the roasted sam-

ple was made up to the weight of the other with pure lead foil, and both buttons were cupelled at the same time, with the following assay-results:

	Ozs. per ton. Gold.	Silver.
Raw ore,	109.6	667.71
Roasted ore,	106.00	629.5
Loss in roasting,	3.60	38.21

The tellurium was determined in this ore and found to amount to 1173.8 ounces per ton; this would give the silver-gold-tellurium compound the following proportions:

	Per cent.
Tellurium,	60.16
Gold,	5.61
Silver,	34.23

showing the mineral to be sylvanite.

One assay-ton of the same ore was placed in an iron retort having an upright inlet-tube and the usual curved delivery-tube. The latter was about 24 inches long, and was connected by 15 inches of glass tube with the bottom of a calcium-chloride tower which was nearly filled with aqua regia. A very slow current of air was aspirated through the apparatus for one hour, during which time the retort was heated to a red heat. During the operation the cover of the retort, as well as both tubes leading into it, became very loose. After cooling the apparatus the ore was taken from the retort and assayed, with the following results:

	Ozs. per ton. Gold.	Silver.
Raw ore,	109.60	667.71
Retorted ore,	100.50	633.15
Loss,	9.10	34.56

Of the above loss there was recovered from the aqua-regia solution 0.03 ounce of gold, 1.10 ounces of silver and 10.87 ounces of tellurium. These results, though they do not quantitatively determine the possible saving of the volatilized metals, conclusively prove the loss of those metals *by volatilization;* had the apparatus been better arranged doubtless results more in accordance with the total losses would have been obtained.

A series of roasting-experiments were next made upon various samples of Potsdam ores as follows:

Name of Ore.	Assay.							Loss per Cent.			
	Raw.			Roasted 20 Minutes.		Roasted 1 Hour.					
	Ounces per Ton.			Ounces per Ton.		Ounces per Ton.		20 Minutes.		1 Hour.	
	Te.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.
Russell (red) kidney ore, with sulphurets	16.7	0.72	58.52	0.64	57.86	0.60	54.90	11.10	1.12	16.66	6.18
Welcome (blue) (Figure 8)	12.95	4.36	16.01	4.30	15.50	4.25	14.36	1.37	3.18	2.52	10.30
Welcome (red)	26.69	1.928	11.172	1.70	10.10	1.51	9.96	11.80	9.59	21.60	10.80
Welcome (blue) sulphuretted	6.17	0.86	5.79	0.82	5.64	0.80	5.55	4.65	2.59	6.97	4.14
Double Standard (red)	5.78	1.22	0.87	1.17	0.85	1.01	0.80	4.09	2.29	17.21	8.04
McPherson (red)	22.04	4.61	5.52	4.56	5.12	3.92	4.72	1.08	7.24	14.96	14.49
Golden Reward (red)	21.58	3.42	3.43	3.30	3.25	3.25	3.22	3.56	5.24	4.97	6.12
Tornado (red)	34.56	3.44	3.36	......	....	3.21	3.14	....	....	6.68	7.44
Ross-Hannibal (blue)	10.24	0.44	255.08	0.41	251.34	......	....	6.81	1.47		
Hardscrabble (red)	56.20	5.20	4.8	5.04	4.61	......	....	3.07	3.95		
Average per cent. of Loss								5.281	4.074	11.446	8.438

Taking an average of the gold, silver and tellurium in the nine ores in the above table (omitting the Ross-Hannibal blue sample, since its large silver-contents make it exceptional) we have:

	Per cent.
Tellurium,	59.97
Gold,	7.64
Silver,	32.39

which answers very well for the composition of the mineral sylvanite. As all but three of the nine ores seem to be completely oxidized, this result indicates that little if any of the tellurium is lost by solution as a result of the oxidization of tellurides.

The average losses in silver and gold, as calculated from the above roasting-experiments, are probably of only approximate numerical value, since they are obtained from the examination of only a small number of samples, which differ widely in mineral character, some samples containing much sulphur and iron as pyrite, much fluorite and some gypsum, others much ferric oxide, some fluorite and very little sulphur; while small amounts of phosphorus and traces of copper were found in some of the ores, one sample contains considerable uranium phosphate or torbernite, and all of the samples carry varying amounts of tellurium, as shown in the table. If the percent-

ages of loss in any pair of ores in the above table are compared it will be noticed that there is much irregularity; the losses seem to follow no fixed law. This, as suggested above, is doubtless a function of differing mineral character, and it is also very likely that unavoidable slight changes of temperature in the furnace, during the roasting, may have had an appreciable effect in this respect. Attention is called, therefore, not so much to the losses numerically reported as to the general fact of a constant, appreciable loss.

Losses by Absorption.

The ill effect of tellurium upon the precious metals at ordinary furnace-temperatures is not evidenced alone by losses through volatilization, but even more forcibly by losses from absorption, whereby the precious metals are carried into the slag to be absorbed by the cupel, pot- or furnace-lining, as the case may be. This effect seems to be due to a partial destruction of the cohesion between the molecules of the precious metals, rendering the metals more fluid and carrying them off, by a sort of entrainment, along with the waste products. The following experiments were made upon pieces of gold-bullion which, by careful assay, contained 83.29 per cent. of gold and 16.91 of silver:

	Au.	Ag.	Ozs. per ton. Au.	Ozs. per ton. Ag.
(1) 29.8 mgms. bullion containing by assay, . .			24.761	5.039
Cupelled with 5 parts of metallic tellurium gave, . . .	20.05	3.639		
Recovered from cupel, . .	3.31	1.365		
Total amount recovered, ...			23.360	5.004
Lost by volatilization, ...			1.401	0.035

Percentage absorbed by the cupel, Au, 13.44; Ag, 27.08.
Percentage lost by volatilization, Au, 5.65; Ag, 0.69.

	Au.	Ag.	Ozs. per ton. Au.	Ozs. per ton. Ag.
(2) 28.45 mgms. bullion containing by assay, .			23.64	4.810
Cupelled with 15 parts of metallic tellurium gave, . . .	14.30	3.025		
Recovered from cupel, . .	8.09	1.701		
Total amount recovered, ...			22.39	4.726
Lost by volatilization, ...			1.25	0.084

Percentage absorbed by the cupel, Au, 34.22; Ag, 35.78.
Percentage lost by volatilization, Au, 5.28; Ag, 1.75.

			Ozs. per ton. Au.	Ag.
(3) 22.17 mgms. bullion containing by assay, .			18.422	3.748
Cupelled with 15 parts of metallic	Au.	Ag.		
selenium gave, ...	10.725	1.875		
Recovered from cupel, . .	5.500	1.200		
Total amount recovered, ...			16.225	3.075
Lost by volatilization, ...			2.197	0.673

Percentage absorbed by the cupel, Au, 29.85; Ag, 32.01.
Percentage lost by volatilization, Au, 11.92; Ag, 17.95.

In each of the above experiments the metals were wrapped in a 12-gramme cornet of pure lead foil, and all were cupelled at the same time.

Microscopic Examinations of Samples from Potsdam Ore-Bodies.

The figures here referred to have been made from photographs of thin rock-sections magnified about 10 diameters.

Figs. 1 and 2 have already been described.

Fig. 3 shows a breccia, highly stained with iron and filled with fluorite. It is greatly oxidized and decayed, but shows a piece of shale and many quartz grains. The section was made from a piece of gold-ore of medium grade, which was taken from the vicinity of one of the mineralizing verticals in the Welcome mine.

Fig. 4 shows an unquestionably igneous rock, from its silica content (76.02 per cent.) a rhyolite. It is a piece of blue, pyritous ore, of good grade, from the Welcome mine. The original rock much resembles that from which Fig. 2 was made, and the sources of the two samples are but a short distance apart.

Fig. 5 shows a bluish, cavernous or cellular quartzite, with no visible pyrite, the cavities of which are lined with druses of quartz crystals. It is from a piece of high-grade blue-ore from the Welcome mine.

Figs. 6, 7 and 8 show quartzites stained with limonite, and cavities lined with druses of quartz crystals. They are all from samples of high-grade (20-ounce), brown Potsdam gold-ores.

Fig. 9 shows druses of quartz crystals and decomposed

pyrite. The quartzes are most beautifully banded, showing layers of growth. This was taken from a sample of light-colored Potsdam ore, of good grade, from Yellow Creek.

Fig. 10 shows a porphyritic rock, surcharged with pyrite; orthoclase is recognizable in numerous large crystals, together with some plagioclase. The ground-mass contains some glass, and is penetrated by numerous veinlets of infiltrated quartz. It is an orthoclase-porphyry. This sample was taken from a piece of dike-rock from Bryant's claim on Squaw Creek. It yielded on assay 1.725 ounces of gold, 7.25 ounces of silver and 32.7 ounces of tellurium per ton. A chemical analysis of this sample yielded:

	Per cent.
Silica,	58.800
Alumina,	13.780
CaO,	8.750
MgO,	.504
K_2O,	9.035
Fe,	4.032
S,	4.693
Total,	99.594

There was also a small amount of sodium which was not determined.

Fig. 11 shows a porphyritic rock containing much pyrrhotite; dark silicates, badly altered by remnants of augite, are recognizable; large plagioclase crystals abound in a finely crystalline ground-mass. The rock is a porphyrite from Two-Bit gulch; it yielded upon assay 2.01 ounces of gold and 5.27 ounces of silver per ton, but no tellurium. The plate is included in this article for comparison with Fig. 10.

Figs. 10 and 11 both represent dike-rocks of very similar appearance to the naked eye; but under the microscope, as in the photograph, Fig. 10 shows secondary mineralizing action in the numerous distinct veinlets of infiltrated quartz, while Fig. 11 does not. In the orthoclase-porphyry of Fig. 10 we find pyrite and sylvanite; in the porphyrite of Fig. 11 the gold and silver are associated with pyrrhotite.

The microscopic examination of the samples here considered shows them to be sandstones partially or completely altered by silicification. Specimens of igneous origin, such as that of Fig. 4, are traceable generally to the contact of the intrusive

mineralizing dikes with the quartzites and sandstones, and, while resembling the latter in carrying gold, owe their rock-character to the former. In cases where oxidizing agents have not acted, mineralized dike-rocks bear the appearance and character shown in Fig. 10; but generally such mineralizing dikes (or "verticals") seem to have suffered oxidation together with a part, at least, of the ores, or impregnated beds, traversed by them.

The mineralization of the altered sandstones seems to be due to metasomasis, as formerly suggested by Dr. Carpenter;* the source of this mineralization, however, to be generically referable to profound rather than to superficial sources, as a secondary result of igneous action. With regard to the ores of the character shown in Fig. 10, which resembles in mineral contents and rock-character the ores of Cripple Creek, Colo., it should be said that few such deposits have been recognized in the Black Hills, but they may yet become of great economic importance.

The Effect of Tellurium on Metallurgical Processes.

Since tellurium has been recovered in every analysis made of the Potsdam ores considered in this article, it may be well to consider the probable effect of that metal upon the saving of the precious metals by the various metallurgical processes in present use upon these ores in the Black Hills; and this review may well include also the question of the saving of the tellurium itself, since there is some evidence that that metal may have commercial value.

Regarding the treatment of telluriferous gold- and silver-ores, we have the article by Dr. E. Priwoznik,† in which he gives the history of foreign experiment in this connection about as follows: In the 50's (as probably now) the tellurides brought to the Royal Smelting-Works, at Zalathna, in Siebenbürgen, were treated in two classes, poor and rich; the one being roasted and the other cupelled with lead direct, in both cases with a large loss of tellurium, silver and gold. Experiments were begun at these works, in 1878, to produce impure tellurium by extraction from the ores with concentrated sulphuric acid.

* *Report Dak. School of Mines*, p. 127. See also his paper, *Trans.*, xvii., 570, *passim.*

† *Ueber das Vorkommen von Tellur.* Vienna, 1893.

The credit of being the first to manufacture tellurium in large quantities is given by Dr. Priwoznik to Councillor Alexander Löwe, who commenced to experiment in 1838, and prosecuted the work till 1854, upon blättererz (telluride of lead and gold with sulphides of lead, copper and antimony) from Nagyag.

His method consisted in first moistening the pulverized ore with dilute, cold hydrochloric acid, to remove carbonates of lime, magnesium and manganese; further amounts of acid were added till effervescence ceased; the ore was then washed, dried and warmed with concentrated sulphuric acid, during which operation sulphurous and hydro-sulphuric acids reacted to form free sulphur. This treatment was carried on in closed, cast-iron vessels, connected through the covers by lead pipes, with a chimney having a good draft. After cooling, the ore was transferred to a lead-lined vat containing water, and stirred with wooden stirrers. Hydrochloric acid was added to precipitate the silver, and in the filtrate from this combination, Löwe precipitated the tellurium, in an impure state, by means of zinc. The residue, containing the lead, silver and gold, was smelted with lead to bullion, and cupelled.

Löwe's work was continued by Max Lill von Lilienbach, who, in 1858, recovered from Siebenbürgen ores about 5 pounds of tellurium, which was precipitated by sulphurous acid, and which sold for about $20.00 per pound.

Methods for the treatment of tellurides were proposed by Prof. H. Schwartz in 1867, and by Prof. F. Stolba in 1870, but received little attention.

In 1871, Anton Schrötter, Director of the Royal Mint at Vienna, began to experiment upon the tellurides from Nagyag. His method consisted in dissolving both gold and tellurium, leaving the silver in the residue. Fifty kilogrammes of rich telluride-slimes were treated with dilute hydrochloric acid to remove carbonates, the mass losing in this operation about 30 per cent. in weight. It was then washed several times by decantation, and then heated with hydrochloric acid till gas ceased to come off. The wet residue was then placed in a porcelain dish, covered with strong hydrochloric acid, and dilute nitric acid was added; after action had ceased, aqua regia was added. The solution was made, and the residue was white. Water and hydrochloric acid were then added to keep the telluric acid

in solution, and the solution was filtered. The filtrate contained the gold, tellurium, selenium, lead, copper, arsenic, manganese, antimony and iron; the silver being in the residue.

To precipitate the gold, Schrötter first used a concentrated solution of ferrous sulphate; later he used oxalic acid, or glycerine, in neutral solution.

The separation of tellurium was next made in the filtrate from the gold, by passing sulphurous-acid gas, without warming and with the addition of water. The addition of water is necessary, because in concentrated solutions a point is reached where further additions of sulphurous-acid gas no longer throw down tellurium.

Attempts seem to have been made to separate the selenium which was always present, by a fractional precipitation with sulphurous-acid gas. Selenium is first thrown down as a red precipitate, and later, the tellurium, which is black.

The tellurium was melted in earthen crucibles (the metal being covered with a thin layer of sodium hydrate) and poured into chalked iron-moulds; the slag was worked over for tellurium.

The quartzose residue from the first operation, containing silver chloride, lead-sulphate and oxy-chloride of antimony, was smelted with litharge and soda.

By Schrötter's method, about 250 pounds of rich tellurium-slimes were worked, containing by smelter-assay about 5.37 pounds of gold and 2.19 pounds of silver, 2 per cent. being deducted from the assay-value of the ores by the smelter from which they were purchased, for probable loss in treatment. Not only did Schrötter's treatment recover the entire amount of gold and silver called for by the original assay, but also something more, together with from 10 to 11 pounds of tellurium, worth, according to the price-lists of Trommsdorf in Erfurt and Schuchardt in Gorlitz, about $90.00 per pound.

In 1874, H. Schnitzler treated some tellurium-ores by Schrötter's method, and found no difficulty in melting, in porcelain crucibles at a low heat and without the addition of fluxes, the tellurium precipitated by sulphurous-acid gas.

In 1876, Anton Hauch, Director of the chemical laboratory at Zalathna, experimented upon the treatment of tellurides. His method consisted in roasting the ores, and catching the

fumes in dust-chambers, whereby about one-half of the gold in the roasted ore became susceptible to amalgamation. The roasted material, containing considerable manganese, was treated in lead-lined vessels with hydrochloric acid, the evolved chlorine dissolving all other metals, but leaving the silver with the residue as silver chloride. From the solution lead and lime were first precipitated as sulphates; the gold was next precipitated with ferrous sulphate, and lastly, the tellurium, by means of zinc. The silver was recovered by smelting the residue with lead. Regarding this process, it is difficult to obtain accurate data, either as to the practical working of these ores by amalgamation, or as to the saving from the flue-dust. The roasting experiment, quoted by Lock and Becker (*ante*), was performed by Hauch. The process does not seem to have received general acceptance, as a modification of Löwe's process has been used by the smelters at Schemnitz. In 1878 and 1879, 110 kilogrammes of impure (73-per-cent.) tellurium was recovered and sold at about $7.25 per pound.

According to a report by Bergrath Johann Dologh, in 1891, Löwe's method was applied in the smelters at Zalathna and Schemnitz about as follows: Into a cast-iron vessel, fitted with a hood, filled with hot sulphuric acid (66 B.) about 1 kilogramme of fine ore was introduced every five minutes, with stirring. For 1 part of ore from 1.2 to 1.4 parts by weight of concentrated sulphuric acid was used, and the mass was heated till it had acquired the consistency of a thick paste. After cooling it was transferred to a lead-lined wooden vessel, diluted six or eight times with water, and allowed to stand twelve hours, with occasional stirring, hydrochloric acid being added to precipitate silver and to dissolve tellurous acid. After settling, the clear solution was filtered into another lead-lined tank, the residue being washed with water, and in this solution the tellurium was precipitated with zinc. To each metric centner of ore 4 to 5 kilogrammes of hydrochloric acid and 30 to 40 kilogrammes of zinc are sufficient. The precipitated tellurium was washed, dried and melted in a porcelain vessel at a bright red heat. The residue, insoluble in sulphuric acid, was mixed with flue-dust or with roasted slimes and smelted with lead for the silver and gold.

In 1893 it was reported that no tellurium was saved at Zal-

athna, all the ores being sent to Schemnitz, where they were treated by a modification of Löwe's method, and three grades of tellurium were recovered, viz.:

28-per-cent. tellurium, which sold at about $3.20 per kilogramme.

40-per-cent. tellurium, which sold at about $4.80 per kilogramme.

60-per-cent. tellurium, which sold at about $9.60 per kilogramme.

The foregoing condensed history of the treatment of tellurides is not inserted on account of any probable value in the methods described for application to American practice, particularly with regard to the Potsdam ores of the Black Hills, since the character of the ores treated and the general conditions under which the foregoing methods were used are widely different from those existing in this country. The matter is interesting, however; and it evidences a persistent and successful attempt to save tellurium for commercial profit.

To return to the subject of the effect of tellurium upon metallurgical processes (the amalgamation of telluriferous ores being omitted from consideration), we may view the matter under two heads:

1. With reference to smelting processes, in which the tellurium itself is lost; and

2. With reference to processes of ore-treatment in which the tellurium may be saved.

In the first case, where telluriferous gold- and silver-ores are smelted with lead, or otherwise, and the proportion of such ores in the charge is relatively small, the losses of gold and silver from volatilization may be lessened or completely avoided, unless the ores in question are roasted previous to smelting, and the losses from absorption may also be reduced to a minimum.

With the Potsdam ores, which are typical dry ores, this would be eminently true, and it might be supposed, therefore, that a smelting-process would be particularly adapted for such ores; but if tellurium itself be valuable, and if it can be saved at a reasonable expense, while avoiding its sinister effect upon the silver and gold, the wet methods in use to-day may be rendered preferable on that account.

Under the second head we have the cyanide- and chlorina-

tion-processes. In the cyanide-process, the cyanide-solutions dissolve tellurium as well as gold and silver; in the precipitation by means of zinc the three metals are concentrated; the coarse zinc is removed by passing the precipitated slimes through a sieve; the slimes are then dried, roasted in a large muffle, treated with dilute sulphuric acid, filtered, washed, dried and melted with fluxes. In one of the newer cyanide-mills working telluride-ores from Colorado, the ores are roasted previous to cyanidation, in order to remove tellurium. The history and effect of the tellurium in the cyanide-process would then be about as follows:

1. If the ore is previously roasted, loss of precious metals and tellurium by volatilization.

2. Solution of tellurium with silver and gold.

3. Precipitation of tellurium with silver and gold by means of zinc.

4. Loss of gold, silver and tellurium by volatilization in roasting the slimes.

5. Further concentration of tellurium, gold and silver by treatment with dilute sulphuric acid; the tellurium not being dissolved.

6. Further losses in gold, silver and tellurium upon smelting, from volatilization and absorption.

In the chlorination-process, as practiced upon the Potsdam ores, the ores, either red or blue, are roasted, chloridized and leached, the chloridized filtrate is treated with sulphurous acid to reduce free chlorine, and with hydro-sulphuric acid to precipitate the gold; the slimes are collected, dried, roasted in muffles and smelted with fluxes.

In this process the history and effect of the tellurium would be as follows:

1. Loss of precious metals and tellurium in roasting.

2. Solution of tellurium, silver and gold by means of chlorine.

3. Precipitation of tellurium and gold in the filtrate by either sulphurous acid or hydro-sulphuric acid and concentration of the two metals.

4. Loss of tellurium and gold by volatilization in roasting the dried slimes.

5. Loss of tellurium and gold by both volatilization and absorption in smelting and cupellation.

It may be truly said that a portion of the losses under (5) may be avoided by reworking all slags, pots and cupels; but that this is a difficult and expensive operation is also true; the slags and rich refuse have a way of accumulating and are difficult to clean.

In the chlorination of the Potsdam ores both the red or oxidized ores (rarely carrying over 3 per cent. of sulphur) and the blue or unoxidized ores (rarely carrying over 13 per cent. of sulphur and probably not averaging over 8) are roasted. This should be, and doubtless is, an outcome of careful experiments, and whether it can be avoided in actual practice remains to be seen. As regards the red ores, the question of roasting or not would seem to be dependent upon the alternative of a finer crushing (with some additional expense) together with increased expense in filtration. The loss then from volatilization in roasting must, for the present, continue. A large portion of the subsequent losses could be avoided by treating the precipitated gold slimes with strong nitric acid, filtration and washing, whereby the gold would be freed from tellurium, second-group metals and other foreign material. From the nitric acid solution, the tellurium could be recovered by evaporation to dryness, solution in dilute hydrochloric acid and precipitation by sulphurous acid or metallic zinc. In the nine ore-samples cited in the roasting-experiments above, there was shown an average content in tellurium of about 22 ounces per ton, and it would seem highly probable that this metal could be saved for a small fraction of its market-value (and a market-value would undoubtedly develop), at the same time avoiding its ill effect upon the saving of the gold.

Fig. I.

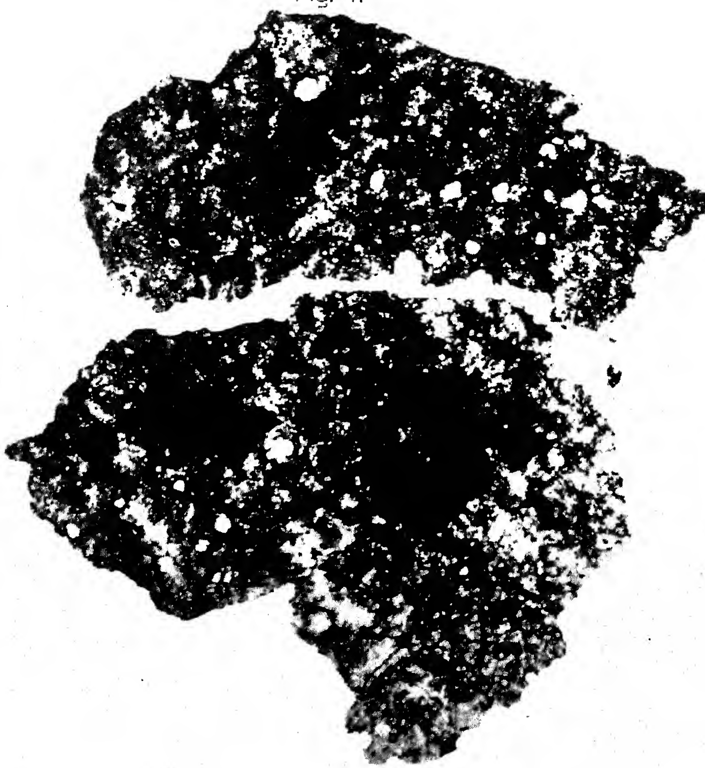


Fig. 2.

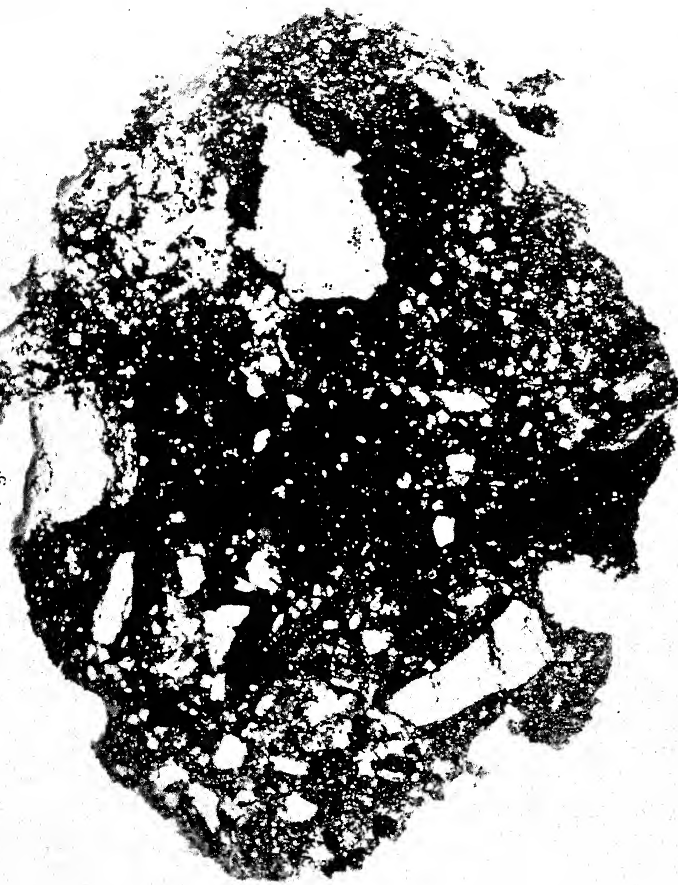
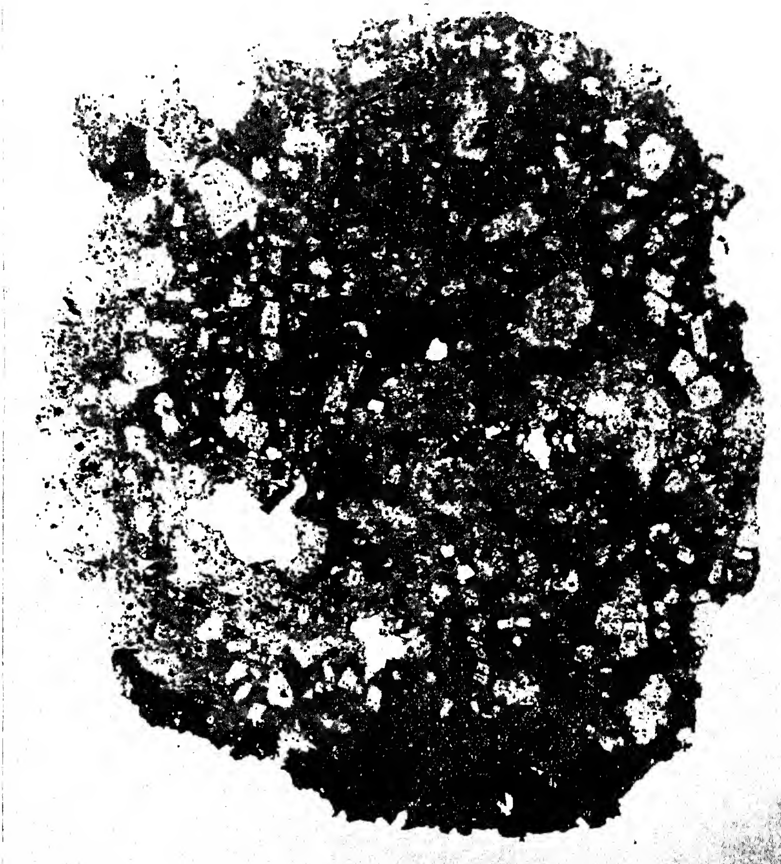


Fig. 3.

Fig. 4.



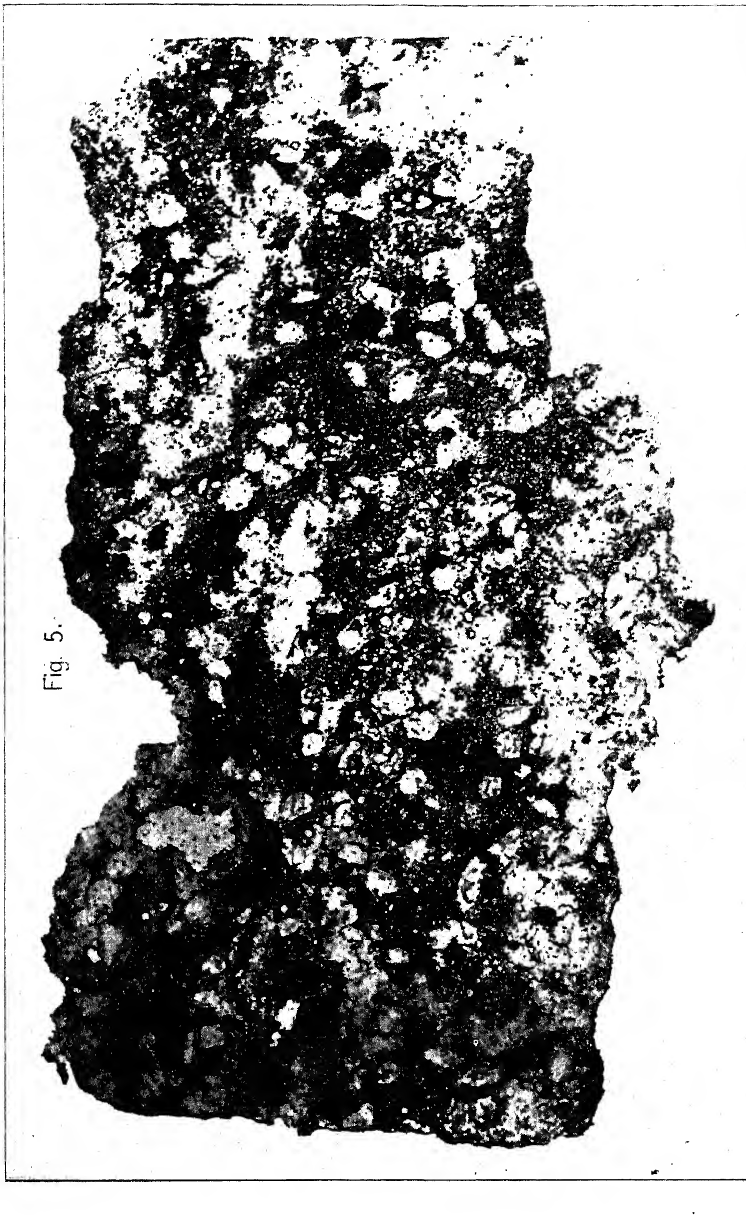


Fig. 5.

Fig. 6.

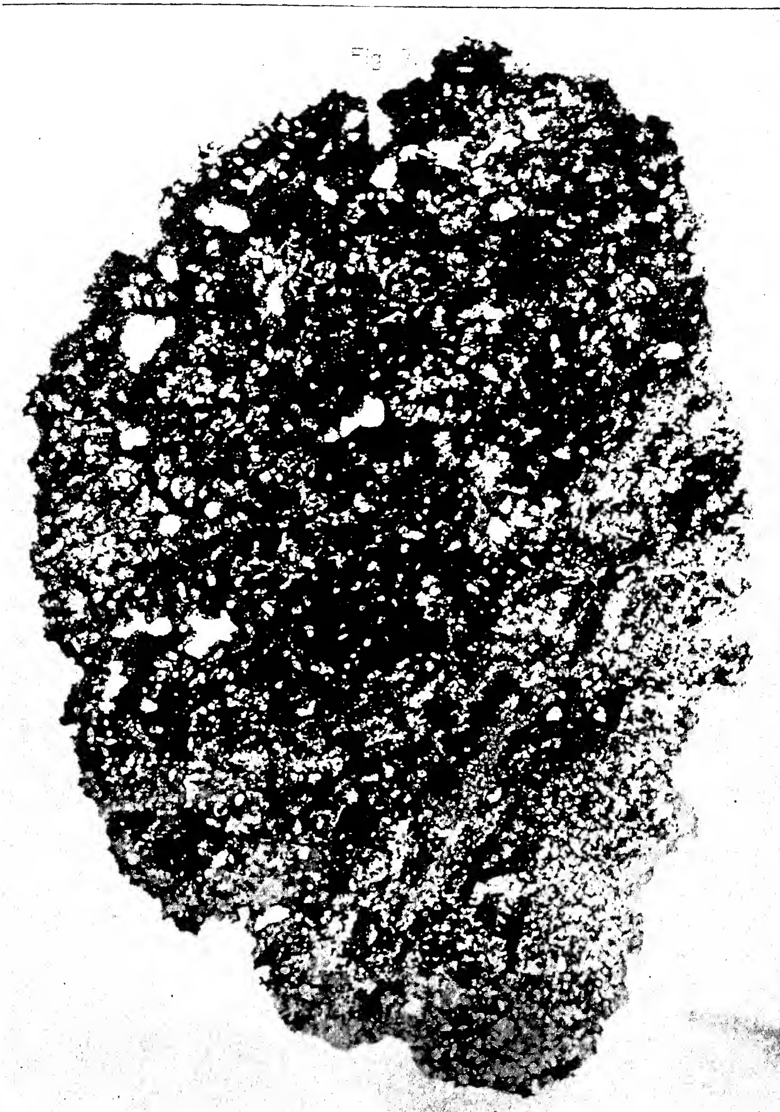
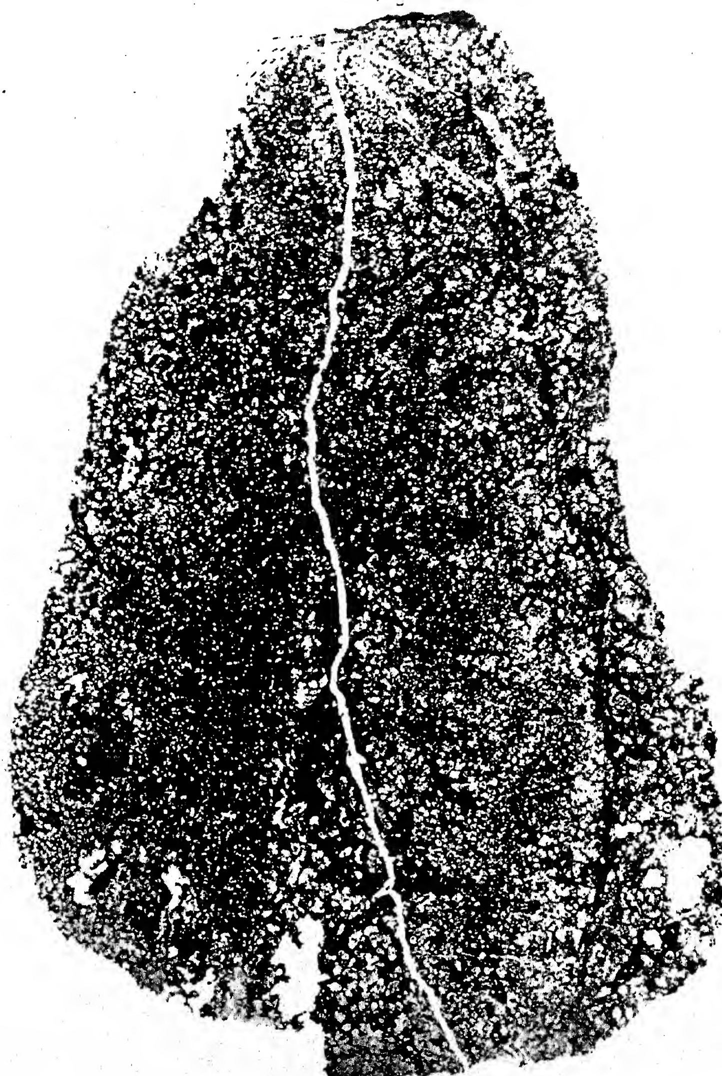


Fig. 8.



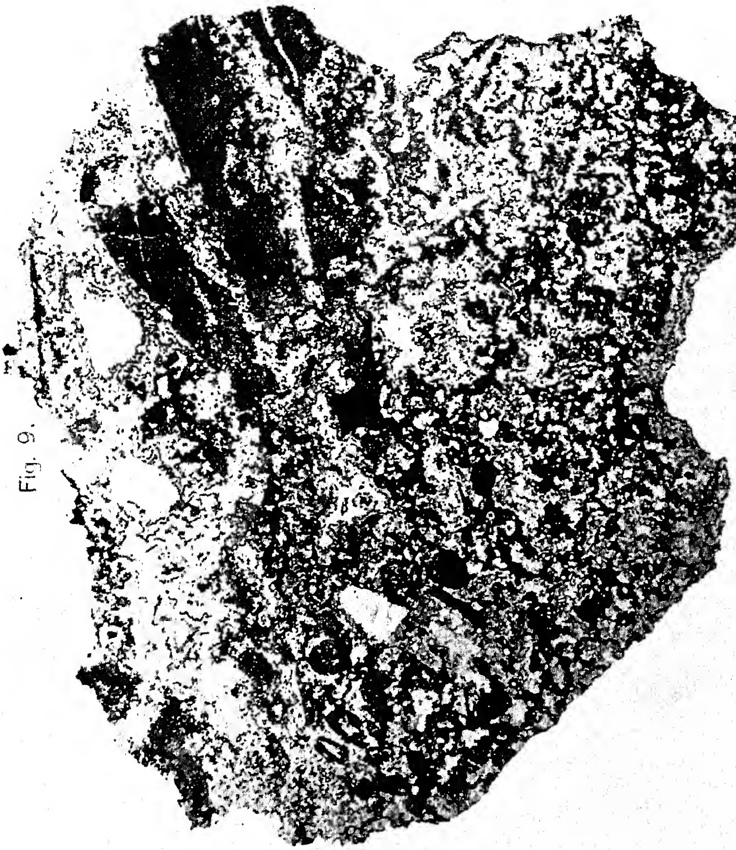


Fig. 9.


Fig. 10.

Fig. 11.

Gold in the Guyanas.

BY HENRY G. GRANGER, BUENAVENTURA, COLOMBIA, S. A.

(Colorado Meeting, September, 1896.)

During upwards of three thousand miles of canoe-travel in South America, including several mishaps, the writer has unfortunately lost his book of notes taken during the year 1894 in Dutch Guyana, or Surinam. But a year spent in a foreign country, visiting its principal mining properties, is necessarily productive of indelible memories and impressions, and some of these may be of interest to the Institute.

The Guyanas include British, or Demerara, Dutch, or Surinam, French, or Cayenne, and the Disputed Territory, or that portion of Brazil on the northeast coast of South America. After passing the picturesque mountainous coasts of Colombia and Venezuela, one reaches a low, flat mud-bank which extends far out into the sea, so that from the steamer, over a couple of leagues of turbid, shallow water, one can just see the dismal outline of the pestilential "bush" that fronts the Caribbean sea, clear around to the mouth of the Amazon. Here, in the order mentioned above, are the Guyanas. So much has been said about this out-of-the-way section during the late excitement over the Venezuela boundary question, that the historical part of the subject may be advantageously condensed.

Gold-mining in the Guyanas is altogether placer-mining, and is of the same general nature throughout; hence by describing the methods in use in Surinam and the results there achieved, an idea will be given of the work throughout the field. The product of this section is about three to four millions of dollars annually, mostly from Demerara, not because of superior richness of its placers, but as a result of its very much larger population, which exceeds twice the aggregate census returns of both Surinam and Cayenne. Mining in all this section is of comparatively recent date, and is indirectly due to the decline of the sugar-industry. After the introduction of beet-sugar scores

of cane-plantations which had been the source of annual fortunes to their absentee-owners were compelled to cease operations, the only survivors being those equipped with vacuum- and other very costly process-plants. For a long time it had been known that in the interior, inhabited only by "bush-niggers" (escaped slaves who live in their native African simplicity) and occasional bands of Carib Indians, there was gold. Attention was, therefore, turned to mining as a means of livelihood; and the industry reached its present development about a dozen years ago.

To the regular course of events there have been three noteworthy exceptions. The first was the "Lava" rush, which occurred several years ago near the head-waters of the Marowyn river, dividing Surinam from Cayenne. Here, amid privation and pestilence and extreme lawlessness, several hundred pounds of gold were taken out in a few weeks; and, as a result of the discovery, permanent placer-workings were established on both the Dutch and the French sides of the river.

Then followed the quartz-excitement in Demerara in 1893 and 1894. The manager of one of the placers discovered a quartz-ledge, a sample of which showed gold. With the lack of engineering prudence which has been characteristic throughout the Guyanas, he immediately ordered a ten- or twenty-stamp mill. After costly transportation, this was finally brought on the ground and erected. The most extravagant hopes were entertained as to its success, and the excitement stirred the entire colony. After just six days' run the entire "gash-vein" had been worked out, and the plant was abandoned.

The third excitement was the "Disputed Territory" rush in 1894. Several negroes, experienced in the placers of Cayenne, made a trip into the uninhabited land across the French border. After their supplies had run out they returned with about thirty pounds of gold apiece. Naturally nearly every able-bodied man in the colony followed them back. The news was flashed over the West Indian cable, and the rush began. For about four months every steamer was crowded to its utmost capacity with men from the West Indies, Central America, and some even from Europe, bound for the new field. Ninety per cent. of the adventurers had never seen a mine of any description. Those first on the ground made a good thing of it; but after the arrival

of the crowd of upwards of five thousand men, of different nationalities, speaking different languages, it was impossible to establish even the semblance of such mining laws as were customary in the early days on our Pacific coast. Labor could not be had except at the generally prohibitive rate of twenty grammes, or about $11.00 per day and "found," the latter item amounting to nearly as much as the wage. When a gang started work on a creek that proved to be rich, others would start in immediately above and below them; and lucky was the man who was not robbed at night of the dust he had panned out in the *batea* during the day. Starvation and fever rapidly did their work; and those who lived were thankful to get away with their experience. This territory is now in dispute between Brazil and France, and regular workings cannot be established until it has been placed under the laws of one country or the other. This having been done, the region will doubtless become fairly productive.

Paramaribo is the source of all labor- and food-supplies in Surinam. Labor has to be contracted before a commissioner, for a period of from three to four months, at the rate of 1.25 guilders, or about fifty cents gold per day, with rations and medicines free. The weekly ration is 6 pounds of rice, 3 pounds of flour, 2 pounds of dried peas, 2 pounds of salt beef, 1½ pounds of salt pork, 1 pound of dried fish, 1 pint of molasses, and ¼ pound of leaf-tobacco. A month's pay has to be advanced to the laborers before they leave town. They are all negroes, speaking a simple *patois*, which is a combination of English, French, Spanish, Dutch and Portuguese, quickly mastered by a foreigner. Some speak also English and Dutch.

The rivers of the colony are the Maro̍wyn, Coremtyn, Commewyn, Surinam and Saramacca, the latter two being the avenues of access to the best producing placers.

Claims are taken up by going to the government office and designating on a map the locality desired, having the quantity of land (not less than 200 hectares*) marked off by scale, and paying a year's tax. For the first three years this tax is 4 cents in gold per hectare, for the fourth year 10 cents, and for

* The hectare is 2.47 acres.

the fifth and subsequent years 20 cents, per hectare. If one desires to prospect new ground a certain number of leagues distant from any working-placer, a large tract may be marked off, free of taxation for the first year. It is against the law to prospect or work ground that has not been regularly taken up. Negroes frequently evade this regulation by taking up a small tract at random, thereby being enabled to pass the police-station, and then working wherever they find gold. If they are arrested on information obtained through rivals, they generally escape by saying that their ground wasn't surveyed and they didn't know where it was. It is well to say here that if a placer-claim has not had a compass-line cut around it under the eye of a government surveyor, intruders cannot be prevented from working right on the land, even though two or more of the boundaries may be defined by streams.

The gold of the colony varies from fine dust to coarse nuggets. One beautiful specimen of the latter has been found, weighing 33 metric pounds, *i.e.*, 16.5 kilos., or about 530 ounces Troy. After local exhibition, it was ruthlessly consigned to the melting-pot.

On both the Surinam and the Saramacca, small open steamers make occasional trips from Paramaribo to the rapids in from three to five days, according to the state of the river. Within the city-limits begins a 12-mile canal, connecting the Surinam and Saramacca rivers. By far the larger part of the several thousand gold-diggers in this colony reach their claims in open row-boats, requiring from six to sixteen days, according to the location of the placers.

The metal is found under the following conditions:

a. In creeks of heavy grade, where the "cargo" is brown hematite-ore, from small pebbles to large boulders. Here the gold is apt to be coarse and is saved by ground-sluicing into a string of from 3 to 6 telescope-boxes, 12 feet long and tapering from 14 to 11 inches. In such workings great care must be taken in the clean-up; for many pieces which seem to be simply iron pebbles and do not catch a particle of quicksilver, prove on fracture to be solid gold with a mere jacket of hematite. The only indication of their value is their specific gravity. In such a manner was found the above-mentioned 33-pound nug get. In such creeks the workable ground is generally con

fined to a narrow strip, which can be worked out with the progress of a single line of sluices.

b. In creeks of less grade, where the "cargo" is quartz and hematite, mixed with granite, syenite and slate. Here the working is by telescope-boxes in strings of from 5 to 8. The head-box is laid on the creek-bed, and the water is sent through the string by a couple of small earthen wing-dams at the upper end. The gravel is then shoveled into the boxes. The fine "cargo" which escapes the fork-man is discharged into an artificial tail-race, kept open by one or more laborers. The workable ground here is somewhat wider than in (*a*), but can generally be handled at one run. The gold will vary from fine particles to small nuggets.

c. In creeks the grade and method of working of which is the same as (*b*), but in which the cargo is entirely quartz-gravel, as sharp-cornered as though just discharged from a crusher. This quartz is mixed with a clay, the consistency of which is such that it will generally disintegrate in from 8 to 12 boxes. The ground is generally wide enough to permit two or three parallel cuttings. The gold is fine, and so much is lost in the first operation that the ground is generally worked over a second, and occasionally even a third, time.

d. In deep flats terminating quartz-creeks (*c*), or occupying a position where there was a level stretch in the course, and the auriferous deposit has spread to a width of sometimes over 100 yards and a length of half a mile. In such there is generally a "dublay" or stripping of from 4 to 8 feet and from 18 inches to 3 feet of gravel beneath. The working here is done by parallel cuts, similar to the Florida phosphate-mining familiar to members of the Institute; only, instead of throwing the "dublay" of the first cut all to one side, it is generally thrown to both sides, and the next cut has either to rehandle it or skip the strip of ground beneath. When grade will allow, and the consistency of the clay matrix is not too stiff, this is washed by a string of sluices into an artificial tail-race. Otherwise it is washed by long-toms. The general long-tom practice is to have one man stripping, one loading and two "chopping" with the long, flexible-handled hoes. The average dimensions of the Surinam tom are: a body 3 by 12 feet; sides 1 foot high, plate with $\frac{5}{8}$-inch holes, at an angle of 45°. Catch-box 4 feet long,

sides 6 inches high and tapering from 3 feet 6 inches wide to 15 inches at the mouth, and having three lath riffles. A little mercury is thrown into the box and above the first riffle in the catch.

e. The banks of the hills bordering on creeks with "cargo" of hematite and quartz, are sometimes workable, either on the surface (in which case the pay will not extend over a depth of 2 feet, and run from \$1.00 to \$2.00 per square yard) or by drifting out a rich clay under a cap of hematite. In the latter case the writer has seen ground ranging all the way from \$5.00 to \$50.00 per cubic yard. The usual mode is to carry the material in trams or barrows and wash in sluices or toms.

In nearly every case the bed-rock is a tolerably stiff clay, often with a distinct line of demarcation between the gravel or pay-clay and the bed-clay. Sometimes, however, the appearance of both the auriferous and barren clays is identical, and only by constant proving with the *batea* can one tell where to stop working. In this bed-clay are occasionally found small but solid layers of hematite, that begin with the termination of the pay and incline at a sharp angle. In the gold-fields of other countries the writer has seen instances of clay-beds with these hematite streaks carrying pay-gravel over them, and, at varying depths underneath, another pay-streak with heavier gold over the true bed-rock; but so far as he is aware, no one in Surinam has ever had the curiosity to sink for the true bed-rock and find what it showed.

The writer has seen a very few cases there, where in working very steep creeks with frequent high falls, the usual clay was not found, and the bed-rock was gneiss.

The accepted rule is, that ground shall be worked as long as it pays 3 grammes (\$1.60) per day per laborer. Experience shows that this is about the minimum limit for making a profit over all expenses. An item of expense that occasionally comes heavily on the operator is sending sick men to town, which, on placers where there is no graduate physician, the law compels after a laborer has been sick in camp three days. The usual sickness is malarial fever, which often proves fatal. The best agent to combat malaria is "Warburg's Tincture," in tablet form, as prepared by standard chemists in both England and the United States. Consumption is by no means rare, and

leprosy is frequently met with, especially among the "bush-niggers." Such is the general fear of the bush-fevers that the owners of some of the principal placers of the colony, though they have become enriched thereby, have never visited their properties.

There have been two attempts made by European genius to work Surinam placers by machinery. In one case a plant was put up that would have been a treasure to any manganese-miner. Loaded cars were elevated to the top of the building, and the material was sent through a long string of sizers and jigs. In the other case, a steam-shovel loaded the gravel into a patent amalgamator, where little jets of water were to keep the pebbles jiggling while the gold got against the copper-plates. Both were flat failures.

The principal placers range from 2000 to 10,000 hectares in area. A few words concerning the average operation will be convincing as to their richness. The object of the laborer is generally to do as little work as he can, and not be sent to camp and fined for refusing duty; and rare indeed is the laborer who will handle over 3 cubic yards per day. The laborer knows that the labor-supply is smaller than the demand; and such a state of affairs has never been an incentive to best efforts. The function of the native foreman is generally simply to see that the laborer doesn't steal, and to take the chief hand in the daily clean-up. The superintendent's or manager's chief duty is to see that the foreman doesn't steal; and the laborer works the placer about as he pleases. It is quite safe to say that in a large proportion of cases the labor accomplishes but little over half what could easily be done with proper direction. A lack of knowledge of the rudiments of mining and business methods is the general characteristic of the operation of Surinam placer-properties. Not one claim in ten has been mapped out, with its creeks and banks and elevations located, and the prospect-value of the workable points noted. The custom is that when a paying creek is found, a camp is started, and work is confined to this one creek as long as there is a particle of pay in it. Then a period of demoralization ensues, while search is instituted for a new pay-creek. Often, when it is found, the cream of it has been already taken by the laborers in night- and Sunday-work.

One placer, under native management, which had been a heavy producer for years, showed a particularly marked absence of method. On one side of a large hill the writer was shown a 3-foot quartz-vein, which, in a pan-test of lumps roughly knocked off, showed free gold. This could be readily traced for half a mile. There had never been an assay of this ore. There was not even the beginning of a shaft sunk on it, although it had been known for a number of years. On the other side of the hill, where there had never been an outcrop seen, the writer was shown an excavation (then filled with water) 10 feet wide, 20 feet deep and 75 yards long. He was told that this was a cut, run in the hope of striking a quartz-vein. A little lower down on the same side he was shown a well-timbered tunnel, into which a span of horses and a top-buggy could have been driven. It was explained that this prospect-drift had been so run in order that, in the event of striking a vein, it could be worked with double-track trains. Thence the writer was taken a little distance to where 40 laborers were running into the hillside an open cut about 100 yards long. It was being run through a very stiff clay shale, and carried on, for the upper part, in steps requiring double handling. Then when the steps got too deep, it was picked down and run out in a tram. This cut was 10 feet wide, and was to be some 50 feet deep at the terminus, and was for the purpose of tramming out some rich gravel that a few prospect-pits had shown on the hillside. In this material four men could have driven an arch-roofed tunnel at the rate of 10 feet per day, that would not have required a stick of timber except to lay the tracks on. To meet such a tunnel with a shaft, and chute the gravel into the trams, would not seem a difficult problem; but such an arrangement had never occurred to the management.

The foreign engineers brought to the colony have generally been one extreme or the other—either the intensely theoretical man, without common-sense or mining experience, two instances of whose misguided genius have been given, or else, what is equally dangerous, the so-called "practical" man, to whom a mathematical instrument is a useless encumbrance, and the tabulated results of careful tests and long experience are a superfluity—the man who "can tell it by his eye." One of the latter class, an English "captain," laid out some work on

the property of an Irishman, who, after thirty years of fruitless rushing to nearly every gold-field in the world, had made his "strike" in Surinam. In the first place, it was a case for drifting, but he laid it out for ground-sluicing. He had to work about six acres of ground with a gentle grade to the north. The bank was from 18 to 25 feet high, and entirely covered with a solid layer of hematite from 5 to 10 feet thick. Not a pit was sunk to prospect the ground ahead. Three ground-sluices were started at the north end. Every afternoon a certain amount of the cap was blasted off; the mornings were spent in breaking up and piling away the products of the evening's blasts, and a few hours' run was made after dinner on the exposed clay. When a sluice had a couple of days' bad run, it was knocked off and started in again further to one side. The water for this operation was stored in a small reservoir, which, at proper grade, could have been supplied from the neighboring creek in a 1500-foot ditch. But the "captain" went three miles up the mountain, and tapped the stream near its source, and brought it down on an average grade of about a foot to the rod. Naturally, about one day in six was spent in repairing the ditch. After everything had been clogged up with blocks of hematite, the work was changed to drifting, but still no prospecting, and the scene resembled rather a ground-hog burrow than the systematic drift-mine that it should have been.

It can be safely stated that no great bargain can be made in buying a Surinam placer. A good mine requiring such simple means to work it as are there successful, is never an elephant on its owner's hands. The only purchase of any note has been a warning example. A number of years ago an energetic foreigner, ignorant of all classes of mining, but bent on making a "deal," brought out a so-called expert, and made a hasty examination of a large property that had been a heavy producer for years. Fabulous amounts were judged to be "in sight," and an option was secured. The property was bought, and a large working capital was provided. The production gradually fell off, and manager after manager, after the failure of his efforts, retired in despair. Finally, a California "practical man" made several ineffectual attempts to restore life by the hydraulic system, and then the property was shut down—the investors being

a quarter of a million out of pocket. The writer saw an elevator put in by the "practical man," a brief description of the remarkable construction of which will possibly be of interest.

With the idea of working a flat of several acres, showing a little gold, a string of sluices was erected with the head-box 12 feet from the bed-clay. At each end of a section of pipe, 7 inches in diameter, a bend was made in opposite directions at angles of about 70°, the upper one resting in the head-box and the lower one on the clay. Aimed at a short distance from the lower mouth of the pipe was the nozzle of a No. 1 monitor, which was connected with piping up the hillside to a pressure of 140 feet. The gravel was to be ground-sluiced to the mouth of the pipe. All was made ready, and both the ground-sluice and the monitor were started. The natural result was that all that was seen of the monitor was a boiling spring of muddy water, and the elevator didn't elevate.

In fine contradistinction to the general rule is the work of Mr. J. J. Schofield, a young Englishman, whose family now reside in Philadelphia, who, in a number of years' practice in the colony as a consulting engineer, has planned and systematized the working of several of the most successful placers.

"Strikes" in Surinam of ground yielding for a time over a pound a day to the ton or sluice, are not rare; and Barnett's yield in 1886 and 1887 of a net profit of over half a million dollars will compare favorably with the record of any placer in any field.

The following are the six principal placers, with their reputed yields and present condition:

Barnett's, ...	$3,000,000.00	Producing.
Green's, ...	1,000,000.00	Producing.
Mueller's and de Joug's, . .	500,000.00	Producing.
Dr. Solomon's, ...	500,000.00	Shut down.
Montana mine, ...	250,000.00	Producing.
Savanna mine, ...	150,000.00	Shut down.

There is an export-duty on gold of 5 per cent.

The question naturally arises, What does the future offer for gold-mining in Surinam? Quartz-mining, with the proper men to develop it, should have a future in this country. Up to the present time it is an absolutely virgin field. A number of veins are known, some of them carrying free gold, but not a

step has been taken to develop this branch of the industry. Regarding placer-mining, the writer is of the opinion that under the present conditions of labor-contracts and transportation of food-supplies, all the ground within available distance that is of any value has been taken up. The great interior, higher and healthier, is untrodden by the prospector's foot. Surinam, though a colony of Holland, has the management of its affairs in its own hands. Its people are intelligent and anxious for progress. It is very probable that any responsible parties who would go to the government with something like the following proposition would meet with acceptance.

Selecting a tributary to, say, the Saramacca river, flowing from the west, which could not be reached in less than, say, sixteen days' canoeing, ask the government for an exclusive concession for forty years (the term of all their mine-grants) for all the territory south of the chosen tributary and west of the river, upon the condition of payment of export-duty for all the gold taken out, a monthly tax of, say, $100, and non-abandonment for over one year. The concession should include the right to introduce laborers from the West Indies (where they are a drug on the market) on contracts of not less than a year, and police authority over them.

During preliminary reconnaissance, a plot of ground could be planted, with the assistance of "bush-niggers," with yams, cassava, okra, peppers, bananas and other tropical vegetables, in sufficient quantity to supply a number of men. A single hunter would keep meat on hand for a company; and, after things were in order, all that it would be necessary to bring in would be salt, rice, coffee, and compressed delicacies for the head camp. Such a venture would cost no great sum, and would very possibly be productive of satisfactory results in both quartz- and placer-discoveries.

[SECRETARY'S NOTE.—Mr. Granger being professionally occupied in the interior of Colombia, it has not been possible to secure from him, down to the date of putting these pages to press, any revision of the proofs of his paper. If corrections or additions should be received before this volume is completed, they will be published in the latter part, under "Discussions."]

Traces of Organic Remains From the Huronian (?) Series, at Iron Mountain, Mich., Etc.

BY W. S. GRESLEY, F.G.S., ERIE, PA.

(Colorado Meeting, September, 1896.)

THE traces of fossils herein described were discovered in or upon piles or heaps of iron-ores upon the docks at Erie, Pa. The author has worked among these ore-piles since 1890, when his attention was first directed to what looked like organic remains upon certain specimens collected. All the specimens described and figured herein were found by him, and, as no iron-ore is used at Erie, and no other iron-ores, save those of the Lake Superior region, find their way to the stock-piles at this place, there can be no question but that the specimens in question are genuine ore transported from that region over the great lakes, and cannot have been brought in from other iron-ore regions. The author has reason to think that no fossil remains have been reported hitherto as occurring in the iron-ores of the Lake Superior region, and he believes that these specimens, however imperfect, rank among the oldest forms of which man has any knowledge.

The illustrations are engraved from photographs of drawings. The specimens themselves have not been photographed, because their condition would have prevented bringing out, by that process of reproduction, all the markings as clearly as is necessary, and would have rendered the specimens otherwise less distinct by reason of the blotchy character of their colors, the presence of cracks, or of marks more or less in contact with those of the organisms. In drawing these remains, therefore, I left out everything upon the specimens of ore that did not seem to belong in any way or contribute to their value as illustrations of the fossils. According to information obtained from the dock superintendent at Erie, Pa., the majority of these specimens came from the Chapin mine, Iron Mountain, Menominee range, Mich. As to the rest of them, I stupidly failed

to note just which ore-piles they were found upon, and was therefore unable to trace them further.

What do these forms and strange markings represent? Unable, though greatly desirous, to answer this question conclusively, the author presents these illustrations to the members of the Institute, in the hope that some explanation may be given by them, and that his discoveries may be followed up by other workers in similar ores, at different places, in order that, if possible, more numerous and better preserved specimens may be secured.

Most of the forms or markings here illustrated occur in the shape of impressions, depressions or channels in the iron-ore; the rest are in relief and are composed of ore. No trace of carbonaceous or animal matter has been observed; the remains are merely casts, moulds or replacements of or by the ore, whatever the original rocks may have been.

With the exception of Figs. 8 and 9, the forms and markings all occur upon planes of lamination (or original bedding or stratification?) and upon flat or undulating surfaces.

Dendritic markings are quite common in some of the lake iron-ores, but are not represented in any of those here illustrated. Nor do any of these markings of organic remains really represent contortions of the laminæ of the rock; nor are we looking at twisted, broken and weathered edges of such laminæ or banding. But that one or two of the figures may, in reality, be mere accidents—nothing more than *formations* of a secondary (?) nature, such as concretionary action might possibly produce, or markings *mechanically* produced, I am prepared to believe. Doubts may also be entertained as to the organic derivation of what is seen in Figs. 2, 3, 4, 5, 7, 8, 9, 15 and 16. Besides those here illustrated, numerous somewhat similar, though perhaps less interesting, markings, such as grooves, running in various directions, and of greater or less dimensions and distinctness, have been detected. I have also discovered distinct rain-pittings, and slates of the more shaly or slaty ore sometimes reveal what remind me strongly of the fossil mouths of vertical worm-burrows, common in the Portage flags of Northwest Pennsylvania. I would mention, also, the discovery of various phases of what seem to have been mud-flows, ripple- and rill-marks, sun-cracks, etc., all in the same ore.

The important fact that one and all these remains (organic and inorganic) unquestionably came from one place or another in one or more of the Lake Superior iron-fields, clears the ground for further observation and investigation in this very interesting field.

A fossil-hunter among these ores has to exercise great caution in order to discriminate between true or *natural* markings on the flatter surfaces of the specimens, and grooves, scratches, etc., produced, in some way, *artificially*. I refer here to numerous markings doubtless produced by abrasion or movement of the lumps of ore upon one another in the mines or in handling and during transportation to docks—markings that are often hard to distinguish from some of those made by organisms in the original sediments. Depressions or grooves that are found more or less filled with ore (often of a different color, texture, etc., from that in which they occur) at once reveal their antiquity and originality, but the very best evidence of genuineness is when one splits a piece of ore and finds the markings upon the newly-exposed surfaces within (see, for example, Fig. 17).

That these discoveries are of some importance, not only to science but practically, the author feels disposed to think. Their practical value may be said to consist in showing that it may not be useless to search for iron-ore in other fields, the rocks of which contain similar fossils. As to the scientific aspect of the matter, the following remarks are offered:

1. The presence of organic remains, or traces of them, in and consisting of ores of iron is not inconsistent with mineralogy, or with the formation of large masses of metallic minerals.

2. Those who would assign to these particular ore-bodies an igneous origin, regarding them as formed of iron, etc., directly derived from volcanic sources, *i.e.*, as primary deposits, ought to explain the presence of organisms in connection with their theories.

3. If we regard these traces of fossils as those of shallow water or near-shore habitat, and even see (as I believe we do) evidence of local areas of dry land in the ore, it becomes difficult or impossible to accept the conclusions of geologists who are inclined to suppose that the original strata (now converted

by processes of concentration and replacement into ore) were deep oceanic deposits, almost beyond the reach of muds and fine sediments.

4. That the characteristic lamination or banding of the iron-ore is, in some places, an original structural feature of the rock, these "fossils" clearly demonstrate.

5. They add something, however small, to our still very scanty knowledge of the fauna and flora (?) of the period to which they belong, be it Lower Silurian, Upper Huronian, or what not.

6. If, by these fossils, the true age or relative geological horizon of the Chapin-mine ore can be determined with something like accuracy, possibly geologists will be able to correlate the several ore-bearing members or series lying west of this mine (perhaps as far as the Mesabi), and eastwardly into Canada, a subject regarding which, unless I mistake, they are not yet agreed.

7. The character of these remains will not support the theory which has been advanced that the iron-bearing strata were originally limestone.

8. They do not suit the precipitation-in-hot-water theory of the ores.

9. I am, perhaps, ignorant of much of the published geological literature of the Lake Superior iron regions; but the discovery of organic remains *in the ore bodies themselves* would seem to demand further and close observation by competent men, so that a rather more rational explanation of the origin and formation of the ore, than has hitherto been given, be, if possible, arrived at. At present, the question appears to be very obscure, because we have, it seems, reached a stage at which microscopic evidence, and perhaps chemical, too, are at variance with palæontology and stratigraphy, and the question that now arises is: Is it for the microscopist and chemist to modify his conclusions to meet the discoveries of the palæontologist, or for the latter to regard his "fossils" as mere accidents—not organic at all—and so let the physicist have it all his own way?

If the petrographer refuses to believe that Figs. 1, 6, 10, 11, 12, 13, 14 and 17 are markings made or left by organisms, then it is for him to give a satisfactory explanation of their nature.

REFERENCE TO ILLUSTRATIONS.

FIG. 1.—Probably remains of marine plants or corals (?), *Oldhamia* (?). Upon a thin layer of softish, fine-grained, sandy, purplish iron-ore. Locality, Chapin mine, Iron Mountain, Mich. Magnified two diameters.

FIG. 2.—Shows roughly-parallel rows of small shallow depressions; some plant (?), or possibly flattened tracks of some crawling animals, upon the surface of a layer or band of red, earthy iron-ore. Locality uncertain.

FIG. 3.—Perhaps casts of remains of plant-stalks. Upon the surface of a fragment of purplish-red ore. Locality uncertain, but the same as that of Fig. 2.

FIG. 4.—Perhaps filled-up sun-cracks. Occurring as ridges on laminæ of a fragment of soft, blue iron-ore. Locality unknown, but the same as that of Figs. 2 and 3.

FIG. 5.—Perhaps the imprint of the cast or mould of a fragment of a marine plant. Upon the surface of softish laminæ of red hematite. Locality unknown.

FIG. 6.—A series of grooved or channeled markings upon the surface of a slab composed of numerous laminæ of softish, reddish-purple iron-ore. Possibly the imprint of the cast of some portion of some marine invertebrate. The two holes on the right may be transverse sections of worm-burrows. Locality, Chapin mine, Iron Mountain, Mich.

FIG. 7.—Perhaps a bit of some marine plant. Composed of dark-red, sandy ore. Locality unknown.

FIG. 8.—Whitish, softish, siliceous material, in reddish inferior ore. Locality unknown.

FIG. 9.—A coral (?) or plant (?). Composed of dull-grayish quartzite, or some other form of silica, etc. Occurring upon a specimen of crumpled, soft, sandy, blue iron-ore. Locality, Chapin mine, Iron Mountain, Mich.

FIG. 10.—Portion (all that was found) of a shallow, sinuous and overlapping channel, the track or trail of some animal (?).* Upon the surface of laminæ in softish, purplish ore. Locality not certain, but probably Chapin mine, Mich.

FIG. 11.—Possibly tracks of some crawling animal; but that on the left reminds me of leaf-scars seen on fragments of fossil coal-plants. Upon the surface of soft, pale-blue, fine-grained ore. Locality, Chapin mine, Mich.

FIG. 12.—Tracks of crawling animals upon nearly flat surface of a slab of bluish-purple laminæ of sandy iron-ore. Note the parallelism of these tracks. The grooves running diagonally across them may have been made by plants scraping over the bottom of the sea or lake. Locality, Chapin mine, Mich.

FIG. 12a.—A few individual foot-prints of Fig. 12, enlarged four times.

FIG. 13.—Possibly plant remains, or animal tracks (?), apparently somewhat side-squeezed. Upon a surface of a fragment of a band of fine-grained, purplish-red iron-ore. Upon the reverse or opposite side of this specimen (which is about ¾ inch thick) are very uniform parallel striæ or

* Compare with the lower left part of Fig. 6. Also see Hall, *Pal. of N. Y.*, .ii, Pl. XVI., and Hitchcock, *Ichnol. of Mass.*, 1858, Pl. XXVI., Fig. 2 (*Unisulcus intermedius*), "strikingly resembling trackway of the common earthworm on mud after a light rain in summer." See also Dr. Walcott's remarks, below.

fine grooves, very suggestive of flattened bark or ribbed plant-structure.* Locality, Chapin mine, Iron Mountain, Mich.

FIG. 14.—Probably track of some crawling animal.† Upon a bedding-plane of a bit of soft, sandy, purplish iron-ore. Locality uncertain.

FIG. 15.—About one-eighth of the surface of one side of a fragment of soft, bluish, fine, sandy iron-ore, exhibiting side-squeezed (?) or distorted animal foot-prints (?), distorted rain-spots (?), or shrivelled plant-remains (?). Locality uncertain.

FIG. 16.—Possibly a bit of a marine plant. Upon the surface of purplish laminæ of ore. Locality, Chapin mine (?), Iron Mountain, Mich.

FIG. 17.—Fragment of fine-grained, moderately soft, grayish-purple, flat, slate-like iron-ore; upon one of the flat surfaces or bedding planes of which are pinkish mottled areas or forms resembling organic remains (?), also composed of fine sandy-looking material. Locality, Chapin mine, Iron Mountain, Mich. Enlarged twice.

Since the foregoing was written, *Science* has published (April 24, 1896, page 622), the following note:

ORGANIC MARKINGS IN LAKE SUPERIOR IRON-ORES.

At the instance of Dr. Charles D. Walcott, Director U. S. Geological Survey, and with the kind permission of the editor of this paper, I beg to submit the following note, hoping that the subject may be brought to the notice of the officers of the U. S. Geological Survey, the Geological Surveys of Michigan and Wisconsin, etc., as well as that of all field workers among the rocks of the iron-ore regions, whose structural and palæontological geology in detail has yet to be unraveled, or is at present being worked up for publication, in this as well as in other countries.

I merely desire here and now to announce the discovery of traces of organic remains, made by me in fragments of iron-ore from the Chapin mine, Iron Mountain, Menominee, Michigan, as well as possibly from other mines on the same range or elsewhere in the Lake Superior region. It is hoped shortly to publish a much fuller account of my work in this connection, in another place.

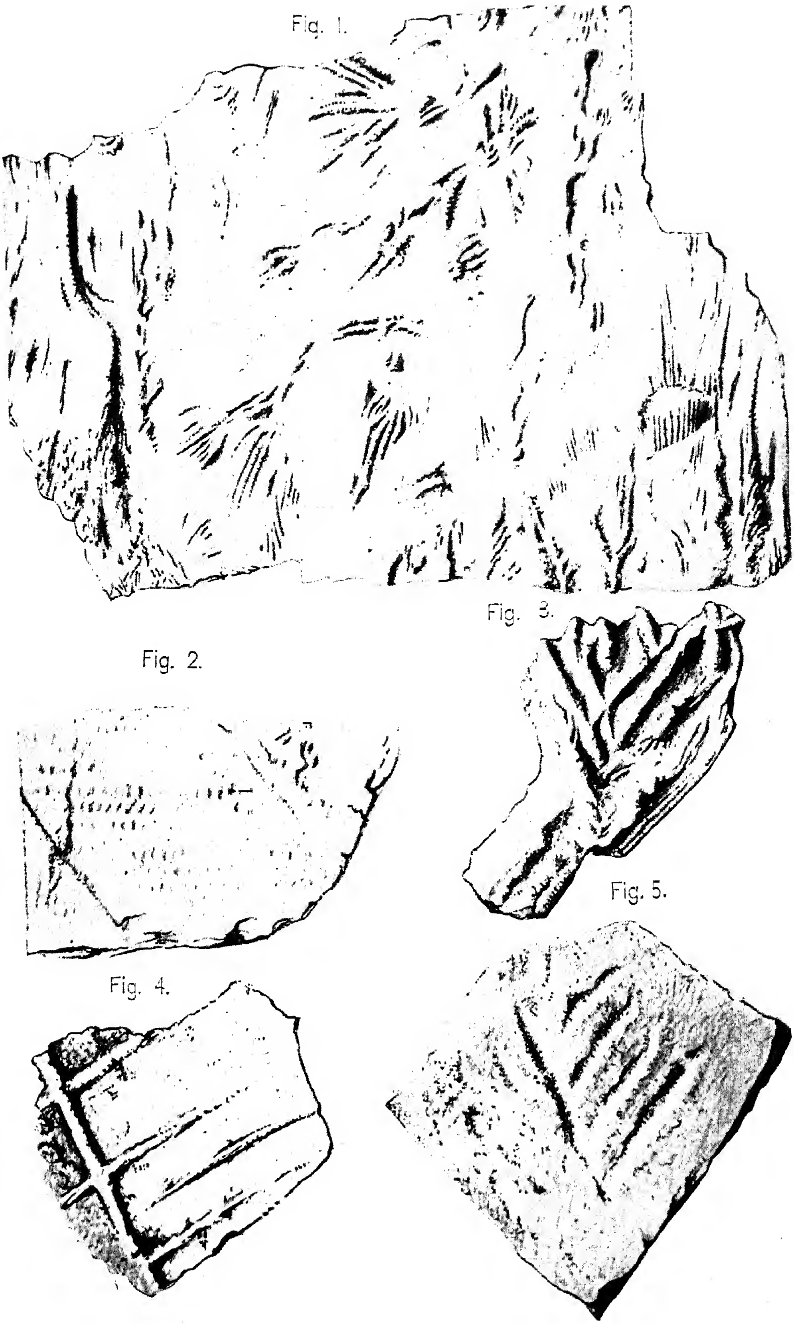
During the period of 1890–93 I collected a considerable number of specimens of iron-ore from the ore-piles on the docks at Erie, Pa., and was firmly of opinion that some of the markings upon them or in them were of organic origin, produced by animals of some kind; but being only an amateur geologist, I decided to submit the material to Prof. H. S. Williams, of New Haven, Conn., for examination. After seeing the specimens, Prof. Williams kindly wrote: "There are certainly some among them which resemble very strongly the trailings left by worms or crawling things on the sand."

The material was then forwarded to the U. S. National Museum, Washington, D. C., where Prof. Charles Schuchert, assistant curator of the Museum—Smithsonian Institution—examined them, and said: "The specimens of the Algonquin ores contain annelid trails."

Finally they were placed in the hands of Dr. Charles D. Walcott for examination, and he kindly reported as follows: "Most of the specimens from the Lake Superior region containing 'traces of organisms in Lake Superior iron-ores' show

* Compare with this parts of Figs. 1 and 2.

† See Dr. Walcott's remarks, below.



Markings upon Lake Superior Iron-Ore.

Fig. 7.

Fig. 8

Fig. 9.

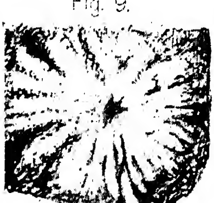


Fig. 10.

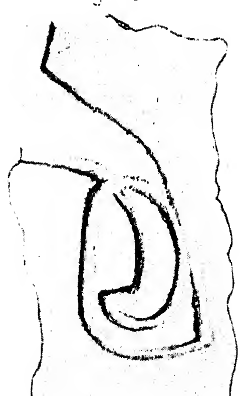


Fig. 11.

Markings upon Lake Superior Iron-Ore.

Fig 12

Fig. 12 A

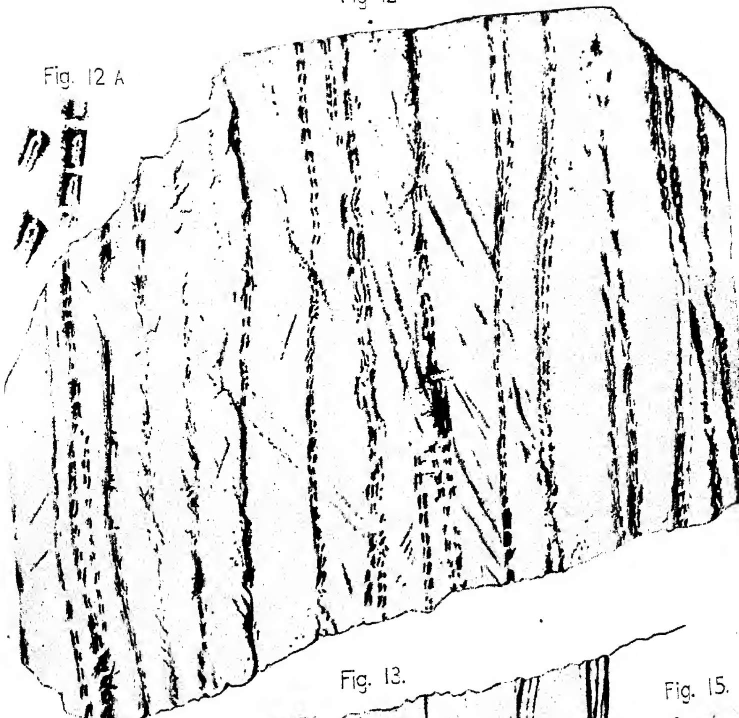


Fig. 13.

Fig. 15.

Fig. 14.

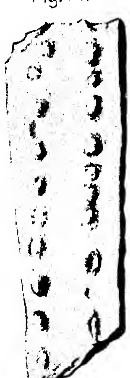
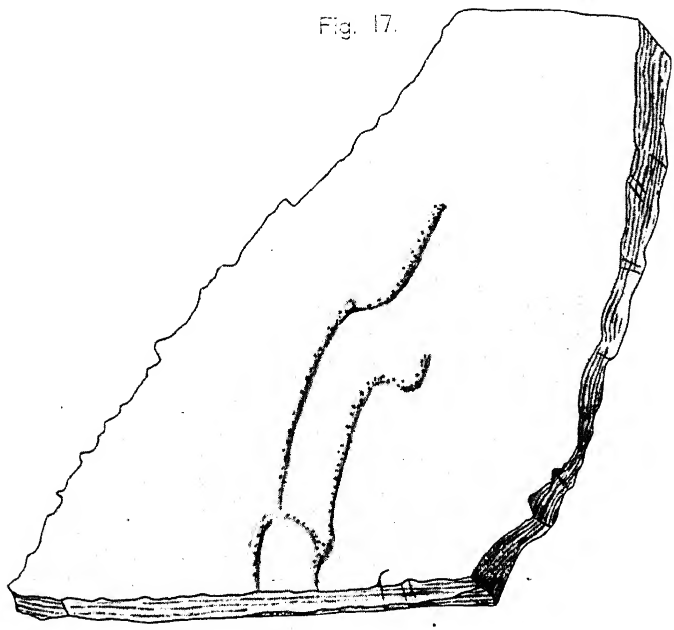


Fig. 16.

Markings upon Lake Superior Iron-Ore.

Fig. 17.



Markings upon Lake Superior Iron-Ore.

NOTE.—In this engraving, the spots upon the supposed organic remains are too definite. These forms are mottled, but not so distinctly spotty as they appear in the illustration.

only markings of mechanical origin. A few, Nos. 10, 14, A, E and G, appear to be casts of the trails of a small annelid, and are, I think, organic. It is not possible to identify them with any described species. For convenience of reference they can be referred to the genus *Planolites*."

Prof. C. R. Van Hise, geologist in charge U. S. Geological Survey, Lake Superior Division, also saw the specimens, and remarks that in his opinion the markings might possibly have been produced by some complex movement or movements, but that they are very peculiar, and in any ordinary case would be unhesitatingly accepted as organic. My long-since-formed opinion as to the organic origin of these markings having thus been confirmed by the highest authorities, this discovery will doubtless add a new phase to the question or controversy regarding the origin and age of the lake-region iron-ores, and iron bearing series of strata, and also should tend to excite renewed and closer investigation of the Huronian rocks in search of better "fossils" than mine, which surely exist and will eventually be brought to light.

Those especially interested could, no doubt, see these specimens on application to Prof. Schuchert, at Washington, in whose care I propose to let them remain for the present. W. S. GRESLEY.

Erie, Pa.

The note in *Science* has led Dr. N. H. Winchell, editor of *The American Geologist*, to remark that it is very important to ascertain, if possible, from which of the two iron horizons at Iron Mountain, Michigan, my specimens came. He says there are two distinct iron-bearing formations in the Lake Superior region, which formations are separated by a very pronounced unconformity. He regards the lower formation as Archæan, and the upper one as Taconic—*i.e.*, Lower Cambrian (?) or even pre-Cambrian.

Dr. G. M. Dawson, Director of the Geological Survey of Canada, says, with regard to my Note, "More traces of life will undoubtedly be found both in the Huronian and Animikie rocks."

Sir J. W. Dawson writes me, calling my attention to his publication, in 1866 (*Quart. Jour. Geol. Soc.*, London, vol. xxii.), of distinct worm-burrows in the Hastings series of Canada, probably Huronian or older. Also to his *Dawn of Life*, 1875, p. 140, as well as to his *Salient Points in the Science of the Earth*, 1894, p. 172.

Lastly, *The American Geologist* for August, 1896, p. 123, has the following editorial comment:

SUPPOSED PRE-TACONIC ORGANISMS.

"The outlook for the detection of organic remains in Archæan rocks is not encouraging. Since the passing of Eozoon, albeit with a vigorous kick, there seems

to have been a panic among the spectral organisms of the crystallines. The Archæan sponge described by Matthew turns out to be an effect of crystallization; other so-called fossils from the Canadian Archæan seem to be largely subjective organisms; and now the radiolarians and sponges, the announcement of whose presence in the crystalline rocks of Brittany elicited wide-spread interest, after an analysis by Rauff, are claimed to be (the radiolarians probably and the sponge spicules at any rate) only mineral aggregates. So it has become evident that the determinations of the palæontologist in this field will no longer pass muster until reviewed by the petrographer.

"Of another sort than these claimed and discredited evidences of life are the supposed annelid trails on the Huronian iron-ores of Iron Mountain, Michigan, recently described in *Science* by W. S. Gresley, and the probable organic origin of some of which has received the endorsement of such excellent palæontologists as Dr. Walcott and Mr. Schuchert. The petrographer has not yet had his say in regard to these things, and though the rock-specimens may show no structural characters which will serve to either affirm or deny the organic origin of these impressions, they will not be admitted as a demonstration of Archæan life without a many-sided scrutiny—the more, as all other similar claims seem to be so rapidly crumbling, and since the Taconic and Archæan ores occur in close proximity in the region mentioned."

Personally, I am very glad to see the *American Geologist* taking up this interesting question of fossils or no fossils in these ancient rocks. In my opinion, what is now needed is diligent and prolonged search among the ores, etc., for better specimens than those hitherto discovered. Nevertheless, I am ready to stake my reputation as an amateur palæontologist that, whatever the specimens of other geologists may show, I am right in affirming that some of the figures illustrating this paper represent traces of organic remains.

Note on Copper in Iron and Steel.

BY R. W. RAYMOND, NEW YORK CITY.

(Colorado Meeting, September, 1896.)

After the publication of the Atlanta paper of Mr. Robert W. Hunt on "Specifications for Steel Rails of Heavy Sections Manufactured West of the Alleghenies,"* I received from Mr. B. F. Fackenthal, Jr., the following comments:

"I note that, in giving the chemical composition of rails, Mr. Hunt does not consider the question of copper. Yet we all know very well that its presence is more or less injurious, as it tends to red-shortness. Bessemer pig-iron carrying

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* *Trans.*, xxv., 653.

copper does not, as a rule, command as high a price as iron which is practically free from copper—a sufficient evidence that this element is objectionable. The iron made from all Cornwall ore will carry at least 0.75 per cent. of metallic copper. I note also that the Cuban iron-ores contain an appreciable amount of copper, say 0.05 per cent.; but as this would not add beyond 0.10 per cent. to the pig iron, the quantity is not sufficient to be injurious. It would be interesting to know, however, why Mr. Hunt and other inspecting engineers do not take the presence of copper into consideration in their specifications and discussions."

To this Mr. Hunt replied as follows:

"The reason that copper is not considered in specifications for the chemical composition of rail-steel is, that we have no evidence to show that its presence has any injurious effect upon the wearing qualities of the steel. It does tend to make the metal red-short during the manufacture of the rails; and this fact, which is recognized by the rail-makers, affects the price they pay for pig-iron containing copper. But I am not aware that it affects at all the wearing-qualities of the metal."

Mr. Hunt's reply was received too late to be included in Volume XXV., and Mr. Fackenthal's question was consequently omitted from the volume. The subject seems to me, however, to be far from exhausted, and I therefore reproduce in this note the brief remarks above quoted, as a possible text for more detailed discussion.

Dr. Dudley, in his famous paper on "The Chemical Composition and Physical Properties of Steel Rails" (1878) says:*

"Almost the only effect that sulphur and copper are known to have on steel is to render it what is technically known as 'red-short;' that is, if a steel rail has too much sulphur and copper in it, it crushes in the rolls or flies to pieces during manufacture. I am unable to find anywhere that sulphur and copper are said to have a deleterious effect on the wear or durability of a rail; and, indeed, some metallurgists claim that they are advantageous in this respect. I have, therefore, not determined the sulphur or copper in the analyses given below, and would not recommend to prescribe any specifications in regard to them. We can safely trust the rail-manufacturers not to give us rails containing too much sulphur and copper."

The practice thus approved by Dr. Dudley seems to have been universally followed ever since, and the latest specifications of Mr. Hunt are silent as to both sulphur and copper. The underlying reason evidently is, that the conditions of manufacture practically guarantee the amount of copper to be too small to injure the rail. What is the actual limit thus

* *Trans.*, vii., 175.

established, is a matter of some doubt. It was formerly assumed that at about 0.4 per cent. of copper, wrought-iron would begin to be red-short; and in 1862 Eggertz* declared the influence of copper on steel to be still greater, so that a steel containing 0.5 per cent. of copper was worthless, while a low-carbon iron would only show, at that percentage, traces of red-shortness. But Wasum showed, in 1882,† that 0.20- to 0.30-carbon steel could carry 0.059 per cent. of sulphur and 0.452 per cent. of copper without red-shortness; and that with 0.06 sulphur and 0.862 copper, or 0.107 sulphur and 0.849 copper, it was "good," giving but slight indications of red-shortness.

Choubley,‡ working with steels containing 0.2 phosphorus, 0.50 to 0.60 carbon, 0.05 to 0.07 sulphur, and 0.36 to 0.54 manganese, found that 0.36 to 0.48 of copper left them entirely weldable at temperatures at and above red-heat. And Mr. Howe§ cites Choubley for the opinion that steel may contain as much as 0.96 per cent. of copper without serious red-shortness, and adds that "Mr. W. W. Scranton habitually makes Bessemer T-rails with 0.51 to 0.66 per cent., which he states are so non-red-short that, in spite of their thin flanges, and the exceptionally low temperature at which they are finished, only from 1.25 to 2.5 per cent. of them are sufficiently cracked to be classed as second-quality."

Mr. Howe gives also‖ an interesting account of the "copper-steel" exhibited by Holtzer at the Paris Exposition of 1889. This material contained copper up to 3 or 4 per cent.; was reported to be red-short when copper had reached 1 per cent.; exhibited in bars a high tensile strength and elastic limit, with considerable elongation. The data furnished concerning it are both incomplete and lacking in precision. But the general conclusion which they indicate is unfavorable to the notion that anything is to be gained in quality by the addition of copper to steel.

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* Wagner's *Jahresbericht*, 1862, p. 9; quoted in Ledebur's *Handbuch der Eisenhüttenkunde* (1895), p. 301.

† *Stahl u. Eisen*, 1882, p. 192, and Ledebur, p. 646.

‡ *Stahl u. Eisen*, 1884, p. 374, and Ledebur, p. 647.

§ *Metallurgy of Steel*, 1890, p. 83.

‖ *Ibid.*, p. 368.

Biographical Notice of Charles A. Stetefeldt.

BY R. W. RAYMOND, NEW YORK CITY.

(Colorado Meeting, September, 1896.)

THE death of Mr. Stetefeldt, which occurred at Oakland, Cal., March 17, 1896, was a surprise, as well as a sorrow, to many of his friends and professional colleagues. In the *Engineering and Mining Journal* of March 28th, I published an obituary article, the substance of which, modified and enlarged by later information, forms the basis of the present biographical notice. I am indebted to Mr. Albert Arents, Alameda, Cal., and Mr. O. H. Hahn, Monterey, Mex., life-long friends, and to Miss Pauline Anacker, of Gotha, Germany, a cousin of the deceased, for details which I could not have obtained elsewhere.

Carl August Stetefeldt was born September 28, 1838, in the village of Holzhausen, in the duchy of Gotha, Germany, where his father was a Lutheran pastor. He was an only son, and (after the early death of a sister) an only child. In his boyhood he is reported to have been physically delicate, but mentally precocious. In 1847 his father was transferred to Hörselgau, near Waltershausen, where, besides discharging his pastoral duties, he kept a select private boarding-school, in which a few pupils received a training preliminary to the "gymnasial" course. Among these pupils (one of whom was the now famous Dr. Knorr, the discoverer of antipyrine) he included his own son, who, thus carefully prepared, entered in 1852, at the age of fourteen, the gymnasium at Gotha.

The records of the gymnasium bear witness that his conduct in and out of school was "blameless," and his attention to study "good." But his career as a "gymnasiast" was marked by an act of rebellion probably more startling then than it would be now. Being much more strongly attracted by the natural sciences than by the dead languages, he conceived a special aversion to Hebrew, and engaged in a conflict with the professor in that department, out of which he seems to have come victorious. At all events, he was not obliged to pursue further

that particular study, and yet left the gymnasium in 1858 with a flattering report of standing and proficiency. It is not an improbable surmise that this incident involved more than the mere protest against a single study. Perhaps the Lutheran pastor had intended for his only son a similar career, so that the study of Hebrew, easy enough in itself, meant also the later study, at the university, of theology. At that period the natural sciences, belonging to the faculty of "philosophy," were still more or less under the ban of the ancient proverb, which declared of the four university faculties, that medicine secured to its votaries bread but no honor; while theology gave them honor but no bread; jurisprudence, both bread and honor; and philosophy, neither bread nor honor! In other words, skilful physicians and surgeons could earn money, but would usually have no official career and rank; pastors would have high social standing, but very small salaries; students of law would find a state career open to them, with possibilities of wealth and rank, while "philosophy" (including everything else, such as languages, history, literature and physical science), was sure of neither profit nor fame. It is true that certain branches of applied science, such as civil engineering, mining and metallurgy, were fostered by the state, and their graduates were taken into government employ; but these formed no part of the university curriculum, being taught in special schools, and by no means exclusively or generally to university graduates.

I think it probable, therefore, that Stetefeldt's successful revolt against Hebrew was, in fact, his choice of physical science as the pursuit and profession of his life. In 1855, while at the Gotha gymnasium, he united with some of the elder students in founding the *Naturwissenschaftliche Verein der Gymnasiasten zu Gotha*, a society of which he was for the first year vice-president, and then president; which has survived and flourishes to this day; and which repeatedly honored him with tokens of its grateful remembrance of his services.

In 1858 he was matriculated at the university of Göttingen, where he remained for two years, studying principally under the distinguished scientific professors, Woehler, Stern, Wilhelm Weber, and Sartorius von Waltershausen. In 1860 he entered the School of Mines at Clausthal, and passed there, in 1862,

the "engineer's examination," receiving in all branches of mining and metallurgy the highest degree. He was at once commissioned to investigate the Pattinson process, and the causes of the "over-poling" of copper at the government works in the Harz.* After visiting Freiberg, Mansfield, Stolberg, and other metallurgical centers, he took the technical management, for a brief period, of a small copper-plant in Bohemia.

In 1863 he came to the United States, and was immediately engaged as assistant by Prof. Charles Joy, then occupying the chair of chemistry at Columbia College, New York. It was in the following year that I made his acquaintance, and he became an assistant in the office and field-work of the firm of Adelberg & Raymond.

This was at the outset of my own professional career, which I began in association with Dr. Justus Adelberg (long since deceased), who had established an office for consulting practice in mining and metallurgy. The time was peculiarly favorable for such an undertaking. There was a great revival of mining industry and speculation, coupled with a great scarcity of trained experts; and the firm of Adelberg & Raymond was so overwhelmed with work as to require the assistance of many young men, mostly graduates of German schools. It is with much satisfaction that I recall the names of many who were thus introduced into American practice, and who have since achieved a professional success to which, as I cannot but believe, their early experience under our direction must have contributed. Among them were Dr. Hermann Credner (now for many years professor at the University of Leipzig and Director of the Saxon Geological Survey), Anton Eilers, Otto H. Hahn, and Albert Arents (recognized authorities in modern metallurgy), not to mention others of less conspicuous distinction. But when I say that, of all that company of young men, Stetefeldt was at that time the most thoroughly and widely accomplished, I believe I express what all recognized. I soon found that he possessed a knowledge of mathematics and chemistry much beyond the usual equipment of a mining engineer or metallurgist, and at the same time an exceptionally wide scientific and literary, as well as technical, culture.

* See *Berg.-u. H.-Zeitung*, 1863, p. 185.

In 1865, in partnership with Mr. John H. Boalt (a Freiberg classmate of mine, and at that time a mining engineer and metallurgist, but now, as the result of a change of profession, one of the leaders of the San Francisco bar), Stetefeldt established at Austin, Nevada, in the "booming" Reese river district, an assay-office and consulting-business, in some respects a branch of our New York firm, but in fact an independent concern. During the continuance of this business he built at Eureka, Nevada, the first lead-smelting blast-furnace erected in that district. Unfortunately, the rich and easily reducible carbonates of Ruby Hill had not yet been discovered. Stetefeldt's furnace was erected to treat the ores of New York cañon, which proved to be not only highly siliceous and refractory, but also scanty in amount. The enterprise was a failure; and when, at a somewhat later period, the Ruby Hill deposits began to supply the materials for a great industry of the same kind, Stetefeldt was already engrossed in another branch of metallurgy, upon which he has indelibly impressed his name.

Soon after coming to America he had taken out a patent for a special arrangement of the Gerstenhöfer shelf-furnace for desulphurizing pyritic ores. This patent, which, considered by itself, was doubtless an infringement, or, at best, a mere subordinate improvement upon that of Gerstenhöfer, was, by agreement with the latter, offered to the American public; but, after a single unsatisfactory trial in Colorado, the enterprise was practically abandoned. According to my recollection the immediate cause of failure in this first experiment lay in the construction and management of the apparatus itself; but back of that was the more serious question, whether the simple desulphurization of auriferous pyrites would leave the material in a suitable condition for the effective extraction of gold by amalgamation. The discouraging answer given by experiment to this question killed the prospects of a good many "desulphurizers" in Colorado a quarter of a century ago; but the history of later years has shown that the practicability of mechanical devices for roasting-furnaces was not thereby disproved.

Stetefeldt pursued this subject with intelligent pertinacity while he was located in Nevada. The silver-ores of Reese river district were treated by preliminary chloridizing-roasting and subsequent pan-amalgamation with the aid of chemicals. For

the preliminary roasting, he first tried a Gerstenhöfer shelf-furnace; but, after repeated experiments, he was forced to abandon this idea, by reason of the caking and incrustation of the ore upon the shelves. The conception of a furnace without shelves followed. I am informed upon excellent authority that this conception was entertained also by the late E. N. Riotte. However that may be, the practical development of it is undoubtedly due to Stetefeldt. Experiments with the old Gerstenhöfer furnace, deprived of its shelves, showed clearly that increased height was required to effect the desired reactions; and working out the details of the apparatus, Stetefeldt built at Reno, Nevada, the first successful furnace of the new type.

The well-known principle of the Stetefeldt furnace is that, especially in a heated atmosphere containing chlorine as well as oxygen, the reactions of chlorination and oxidation upon particles of ore sufficiently small and sufficiently exposed occur with such rapidity that a free fall through such an atmosphere, coupled with the subsequent period of repose at the bottom, will suffice, without further manipulation, to produce the desired result. The invention was undoubtedly both a novelty and an improvement; and though the practical limits of its advantageous use are still matter of controversy, concerning which I express no opinion here, there can be no question that it has given to Stetefeldt a high and permanent place in the history of metallurgy.

This reputation he earned by labors far exceeding those of the mere originator of a happy thought, thrown out for others to perfect and apply. Indeed, I am inclined to think that too much credit is popularly given to the geniuses who conceive bright ideas, and too little to the experts who reduce them to successful practice. We hear, perhaps, too often of the claims of this or that inventor to priority in the desire, intention or attempt to do a thing which he did not accomplish, and which other brains and hands subsequently developed to practical usefulness. It is especially noteworthy, therefore, that Stetefeldt remained, to the day of his death, the leading authority on his own furnace, the details of its construction, the method of its operation and the sphere of its application. He was not only a chemist and metallurgist, but an engineer; and the fearless confidence which he reposed in his own knowledge and experience enabled him to deal with many a troublesome problem of

practice before which the ordinary inventor would have stood helpless.

In 1870 he went to Europe; but his father had died, the friends of his youth were scattered, and the ties that bound him to the New World were stronger than those which remained to hold him in the old. He returned to San Francisco in 1872, resided there until 1882, and then in New York until 1889, after which he remained, until his death at San Francisco, residing in Oakland, the pleasant city across the bay. He was, however, much away from home on professional business, which concerned chiefly the construction and operation of his furnace and the development of metallurgical operations to which it was auxiliary. The most important of these was the Russell lixiviation-process, which he did much to improve and to recommend, publishing on the subject several papers, and a text-book, the second edition of which appeared last year. One of his latest enterprises in connection with the improvement of silver-mines was the introduction of a producer-gas for firing the dry-kilns and the roasting-furnace at the Marsac mill, Park City, Utah—a new departure which promises to be of great importance.

Mr. Stetefeldt did not join the American Institute of Mining Engineers until 1881; but, once a member, he took an active interest in the society, and enriched its *Transactions* with some of the most valuable papers they contain. The list of his contributions is as follows:

VOLUME.	PAGE.	TITLE.
XII.,	95.	—The Shelf Dry-Kiln.
XII.,	291.	—Remarks on the Leaching of Silver-Ores.
XIII.,	47.	—Russell's Improved Process for the Lixiviation of Silver-Ores.
XIII.,	309.	—Remarks on Pressure-Filters.
XIII.,	369.	—Notes on the Patio Process.
XIV.,	336.	—The Amalgamation of Gold-Ores, and the Loss of Gold in Chloridizing-Roasting, with Especial Reference to Roasting in a Stetefeldt Furnace.
XV.,	355.	—Russell's Improved Process for the Lixiviating Silver-Ores, in Its Practical Application.

VOLUME.	PAGE.	TITLE.
XX.,	3.	—The Construction of Details for a Modern Lixiviation-Plant.
XX.,	15.	—The Precipitation of Metals from Hyposulphite Solutions.
XX.,	37.	—The Refining of Sulphides Obtained in the Lixiviation-Process with Hyposulphite Solutions.
XXI.,	74.	—Experiments with the Roessler Converter at the Marsac Refinery, Park City, Utah.
XXI.,	286.	—The Marsac Refinery, Park City, Utah.
XXII.,	659.	—Remarks on the Capacity and Effectiveness of the Stetefeldt Furnace.
XXII.,	724.	—Remarks on the Specific Gravity of Gold in Gold-Silver Alloys.
XXIII.,	134.	—The Consumption of Fuel in the Taylor Gas-Producer Plants at the Aspen and Marsac Mills Compared.
XXIV.,	3.	—The Stetefeldt Furnace.
XXIV.,	221.	—Product and Economical Results of the Marsac Refinery for the Year 1892.
XXIV.,	530.	—The Inaccuracy of the Commercial Assay for Silver, and of Metallurgical Statistics in Silver-Mills, with Special Reference to the Treatment of Roasted Ores by Amalgamation and by the Russell Process.
XXIV.,	573.	—Note on the Taylor Gas-Producer Plant at the Ontario Mill.
XXIV.,	868.	—Remarks on the Inaccuracy of the Commercial Assay for Silver.
XXV.,	993.	—Remarks on the Lixiviation of Silver-Ores.

On page 443 of Vol. XVIII. an abstract is given of a method proposed by Mr. Stetefeldt for the treatment of auriferous tellurides. His full account of this method will be found in the *Berg- und Hüttenmännische-Zeitung*, 1865, vol. xxiv., p. 374.

In recognition of the value of these contributions and of his

eminence in metallurgy, Mr. Stetefeldt was elected in 1888 a vice-president of the Institute; and in 1895 he was again elected to that position, which he occupied at the time of his death.

As Secretary of the Institute, and editor of its *Transactions*, I must bear witness to the unusual excellence of Mr. Stetefeldt's English style. Not only do few foreigners express themselves in English with such terse and clear precision; the accomplishment is all too rare among native writers on technical subjects. And the moral I would draw is one upon which I have often elsewhere insisted, namely, that university culture is sure to tell, even in a special professional career. Other things being equal, the man who has it will surely come to the front.

Of Mr. Stetefeldt's numerous contributions to scientific literature, elsewhere published in Germany and in this country, I have no record; but I know that they were numerous, and that they dealt with branches outside of his special profession, as well as those with which his name was chiefly connected. Heliology and selenology, for instance, were subjects which he had studied with special interest, and on which he speculated with ingenuity as well as learning. A few months ago he published an excellent English abridgment of an elaborate paper by Prof. Suess,* in which the phenomena observed upon the moon's surface were made to yield interesting and important suggestions concerning the eruptive rocks of the earth.

Mr. Stetefeldt was married December 31, 1872, but had no children; and his wife died some years before him. His own death was the result of a disease of the heart, from which he had suffered periodically for several months, and which was probably aggravated, towards the last, by a mistaken resort to outdoor exercise. When informed that he could not recover, he refused to take any more medicine, saying, "Let me die in peace." We who lament his departure can but echo these pathetic words, as we say above his grave, *Requiescat in pace!*

* "The Moon as Seen by a Geologist," *Publ. of the Astron. Soc. of the Pacific*, vol. vii., No. 42, page 139, June, 1895.

*

The Use of the Tremain Steam-Stamp with Amalgamation.

BY EDWIN A. SPERRY, GUNNISON, COLO.

(Colorado Meeting, September, 1896.)

THE use of steam-stamps in the crushing of ore for the purpose of amalgamation has been very limited, and little has been written on the subject. As the writer has been operating a mill of this kind during the past year, it is possible that he may be able to present a few points in regard to mill-practice in this line which will be of interest.

The mill herein referred to contains two Tremain steam-stamps, a more detailed description of which will be given below. A bond and lease had been taken on a property on Cross mountain, Gunnison county, Colo., and the parties operating wished to prospect it thoroughly and did not wish to ship out the ore, owing to high freight-rates. They looked around to find some mill that would answer the purpose and not be too expensive. The steam-stamp in question was finally decided on, and the writer was employed to erect and operate it.

The economy of construction was demonstrated at the outset. The stamps were dropping within ten days from the time that the machinery was unloaded on the ground. A large portion of this period was consumed in waiting for supplies and fittings which should have been on the ground at the same time with the machinery.

Ore.—The ore to be treated was a mixture of white, porous quartz, with the decomposed and undecomposed oxides of iron and manganese in varying proportions, from the pure white quartz to solid pieces of the oxides. It appeared to be the product of replacement in a soft dolomite at or near the line of contact with a dike of porphyry. On careful examination with the aid of a microscope it was found that the gold was in the form of an extremely thin film over the surfaces of the quartz, making only a bronzy-yellow coating, quite readily distinguished from the film of iron oxide, which was also present.

Mill.—The mill was constructed on the usual plan, with the ore-bin above and against the hill, which was cut into for the

main building. The bin was of about 20 tons' capacity. No rock-breakers were used, but the larger pieces of ore were broken by hand to a size that would pass through a 3-inch ring. The ore passed into a Challenge ore-feeder, from which it was fed into the mortar.

For the benefit of those not familiar with the steam-stamp under discussion, although some may be, a description will be given so far as to make clear any reference made hereafter to any part.

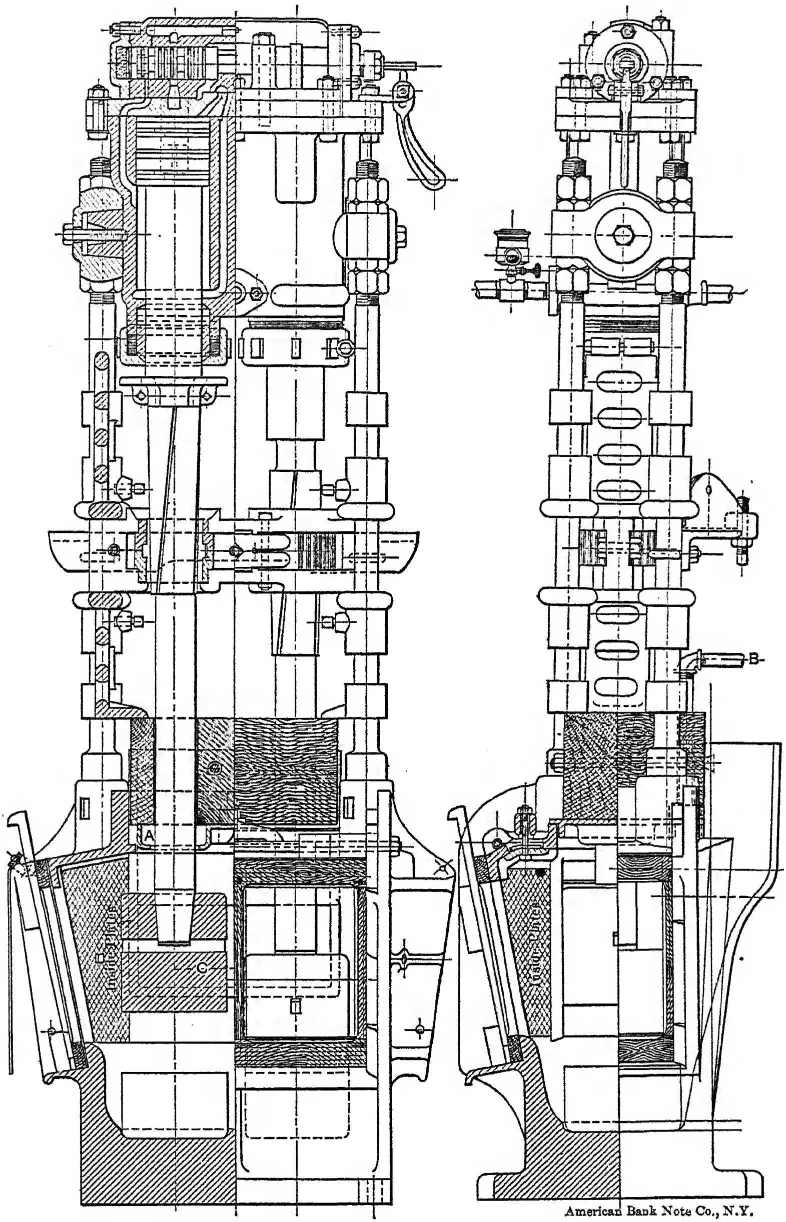
The sections of the stamp shown in Fig. 1 will give some idea of its mechanical construction.

The mortar is 12 by 20 inches in size at the inside of the lip of the discharge, and 14 by 24 inches at the outer edge of the lip. Screens are placed on three sides, the front and the two ends, giving 384 square inches of screen-surface. The depth from the discharge to the bottom of the mortar, including the recess made to receive the die, is 7 inches. The die, being 5 inches high, leaves only 2 inches issue for the pulp. This was increased to 6 inches by the addition of a 4-inch "chuck-block" carrying a copper plate on the inside. This reduced the screen-area to 256 square inches, but did not materially decrease the capacity.

The stamp comprises the piston, stem and stamp-head, and, when newly shod, weighs 300 pounds. The steam lifts the stamp by a 6-inch piston-head, the lower area of which is reduced by the 4-inch stem, which extends downward from it, leaving, as a lifting-area an annular surface 1 inch wide at the outer edge of the head.

When the piston is raised to a certain point, connection is made between the two ends of the cylinder. The upper surface of the piston-head being greater, the stamp is forced down with a force proportionate to the surfaces, together with its weight, giving a blow of about the same force as that of a 1000-pound stamp. The length of the drop varies from 5 to 8 inches, according to the amount of wear on the shoe and die. Provision is made for the operating of the automatic feeder, by cutting a recess in and around the stem near the steam-chest, into which a collar is bolted which actuates the feeder-arm. Reference will be made to this later on. The shoes are cast similar in form to the gravity stamp-head, with the exception of the recess

Fig. 1



Tremain Steam-Stamp, Vertical Sections.

in the lower end, to receive the shank of the shoe. They are fitted to the stem by placing short feathers of sheet brass around the sides of the hole in the upper end of the shoe, placing it on the die in the battery, in its proper position, and allowing the stem to drop into it. The entire stamp is raised with a bar and allowed to drop several times until secured. Steam is turned on slowly and the stem driven to place by a few blows, when it is ready for regular work. There is a system of ratchets and pawls which revolve the stamp on the upward stroke, leaving it to fall in a direct line. There is, on an average, 1 complete revolution to about 15 strokes.

The shoes and dies wear very evenly and smoothly, and with the ore in question wore about ½-inch each per 100 tons crushed.

Thirty-mesh screens were used at first, but on experimenting, it was decided to use 20-mesh instead. This gave a product of the following description, as to proportions of size and values. The crushing is evidently fine enough, and could be coarser but for the fact that the tailings were to be subjected to a subsequent treatment, which required the size to be no coarser.

TABLE I.—*Results of Screening.*

Size.	Per Cent.	Value Per Ton.		Contents, Per Cent.	
		Gold.	Silver.	Gold.	Silver.
On 40 mesh............	28.5	$14.00	$1.12	18.07	26.7
" 60 "	4.0	13.00	0.72	2.36	2.4
" 80 "	10.5	14.00	1.00	6.65	8.8
"100 "	8.5	16.00	0.92	6.16	6.5
Through 100 mesh...	48.5	30.40	1.37	66.76	55.6
	100.0			100.00	100.0

The mortar, as originally designed, was intended to have four amalgamated copper-plates inside. These were to be placed in the corners and were very small. The splash was so great that they scoured badly. They were covered with wire-screen cloth, and the results were greatly improved, but they were finally abandoned entirely, and the copper-plates on the "chuck-blocks" were substituted, but these were not satisfactory. Very little amalgamation is accomplished in the mortar; fully 95 per cent. of the amalgam recovered comes from the outside plates. Of

these there are 8, of which 3 are attached to the mortar, one just below each screen. These are cleaned up but once a week. There are 5 apron-plates, $4\frac{1}{2}$ feet wide, and 16 feet in aggregate length.

The first is 8 feet long; and the second, third, fourth and fifth are each 2 feet long, and are arranged as shown in Fig. 2. This arrangement, in this case, is very effective, as the mercury in the two wells can be kept active by the addition of sodium amalgam, preventing in a large degree the loss of quicksilver "sickened" by the oxides in the ore. By this method the loss of mercury was reduced $\frac{1}{2}$, or from 2 ounces to 1 ounce per ton of ore treated.

Owing to the fact that the proportion of heavy oxides was very great, sometimes amounting to 60 or 70 per cent., it was quite impossible to use the common forms of traps to recover mercury, and two forms were finally adopted which proved quite satisfactory. One was in the form of a "clean-up pan," which was arranged to be continuous in feed and discharge, and constantly stirred. This was used outside of the mill. The other was of the form shown in Fig. 3.

This was effective and convenient, saving considerable "floured" and "sickened" mercury. It is a modified form of the *Spitzlutte*, having a copper-plate on the slope on which the pulp falls. In the figure A is an inlet-pipe to supply water for the upward current at B. C is a board set in across the trap to prevent surface-currents. D is a V-shaped trough across the bottom of the trap to receive the mercury and the heaviest sands. E is an opening into the trough at its point, into which is screwed a short nipple, covered with a cap. When it is necessary to clean the trough, the cap is taken off from the nipple and a piece of pipe which fits closely inside the nipple is inserted and worked in and out, cleaning the entire length of trough. The copper-plate on the sloping bottom of the trap worked well, but it was necessary to clean it often, owing to filming by the oxide.

Wood.—About $\frac{1}{10}$ of a cord of dry spruce and pine was used per hour, or about $\frac{2}{10}$ cord per ton of ore. This would indicate about 12 H. P. as the power required to operate the stamps. This could be greatly reduced by covering the boiler with a brick casing.

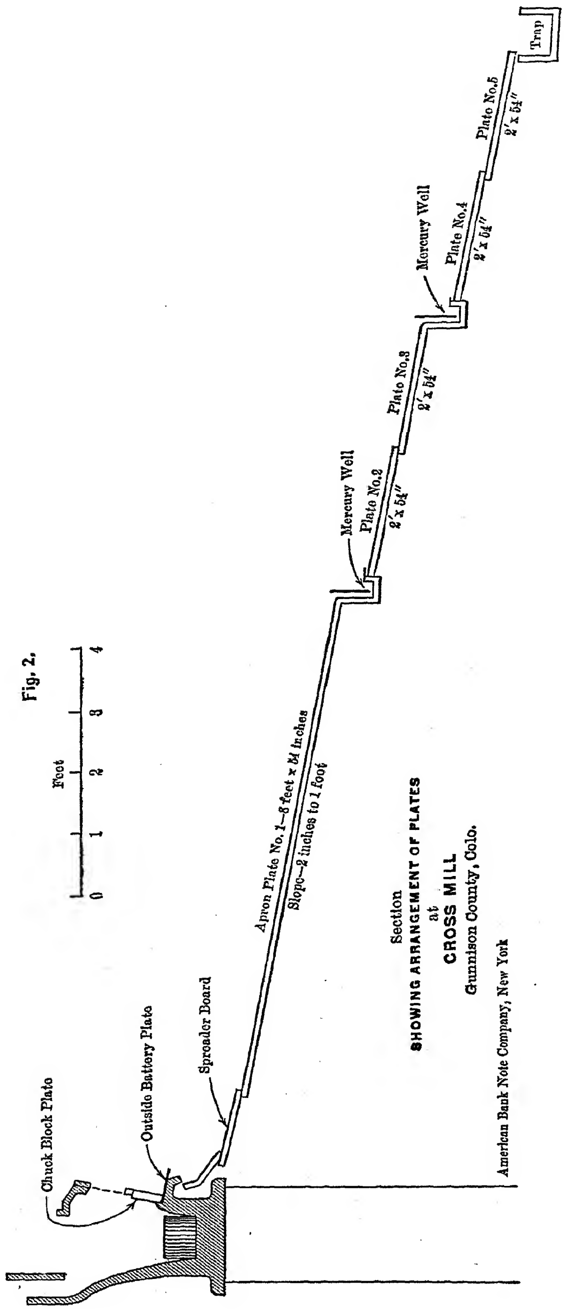


Fig. 2.

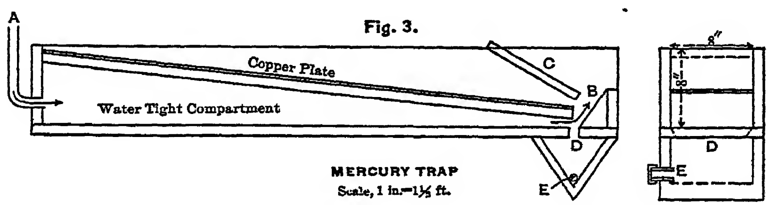
Section
SHOWING ARRANGEMENT OF PLATES
at
CROSS MILL
Gunnison County, Colo.

American Bank Note Company, New York

Capacity.—The capacity, of course, varied according to the nature of the ore. Sometimes it amounted to $\frac{8}{10}$ of a ton per hour, where the ore was soft and decomposed. At other times it was only $\frac{4}{10}$ of a ton per hour, when the ore was principally quartz. The average was 12 tons per 24 hours.

Water.—The apron-plates were at first set on a slope of $1\frac{1}{2}$ inches to the foot, and, owing to the large amount of iron present, it was necessary at times to use as much as 3000 gallons of water per ton of ore. On setting the plates on a slope of 2 inches to the foot, this was reduced to 1000 or 1200 gallons per ton. The water supply is furnished by a $1\frac{1}{4}$-inch pipe line about 500 feet long, with a head of 30 feet, which is more than sufficient for all needs.

Fig. 3.



General Remarks.—Considerable objection has been offered by some based on the alleged liability of oil or oily water finding its way to the plates from the steam-chest. At first, this did annoy us; but by keeping the packing of the piston in good condition, it was almost entirely avoided. As a precautionary measure, we substituted for the collar or flange attached to the stem to actuate the feeder-arm, as before explained, castings of similar form, only with the addition of a rim around the edge to form a cup. This cup was of sufficient size to extend out from the stem the same distance as did the tightening-rings of the steam-chest, so that any drip would fall into it. This was filled with cloth, which would absorb the drip as it fell, and was occasionally sprinkled with soap shavings or pearline, which emulsified the oil carried down. Graphite lubricator, manufactured by The Joseph Dixon Co., will probably soon be experimented with, and the results will be reported. A circumstance may be mentioned in this connection which may be of interest. At first, lye or caustic potash was used to destroy the oil, and a greenish scum would form on the plates which it was impossible to keep off. On entering the mill one day, a

faint odor of ammonia was detected, and the idea suggested itself at once that there might be sufficient iron held in solution by the battery water, dissolved from the ore, to be acted upon by the ammonia generated by the chemical combination of the sal-ammoniac (used to clean the plates occasionally) with the caustic potash, and be precipitated as ferrous hydrate, $Fe(OH)_2$, forming the scum mentioned. Acting on this suggestion, samples of the water were filtered off from the tailings and treated with NH_4OH. The hydrate was precipitated in considerable quantity. As a check, other samples of the water were treated with K_4FeCy_6, and, after standing a few hours, the blue coloration was very strong. The use of lye was discontinued, and the scum disappeared.

After about 400 tons of ore had been crushed, a scale was formed on the large apron-plate (No. 1) composed of hard amalgam, making the plate work unevenly. This scale was removed, and the value of it was found to represent about 10 per cent. of the entire value of amalgam recovered from the plate, or about $200, or 50 cents per ton of ore treated. Tests were made to determine the fineness in gold and silver of the bullion produced from this scale, as well as from the amalgam from the several plates, taking the three outside battery-plates as one.

The results were as follows:

TABLE II.—*Fineness of Bullion from Scale and Plates.*

Product.	Scale.	Battery Plates.	Apron Plate No. 1.	Apron Plate No. 2.	Apron Plate No. 3.	Apron Plate No. 4.	Apron Plate No. 5.
Amalgam (mg).........	5000	4000	1860	4000	2000	2000	3000
Retort (mg)..............	1607	1310	413	525	307	260	426
Retort (per cent.).....	32.1	32.8	22.2	13.1	15.3	13.0	14
Gold (1000ths)..........	840.9	827.7	781.2	575.2	566.6	555.8	547.5
Silver "	159.1	172.3	218.8	424.8	433.4	444.2	452.5
	1000.0	1000.0	1000.0	1000.0	1000.0	1000.0	1000.0

The bullion was refined directly from the retort of the different samples, as the proportion existing between the gold and silver was all that was sought. The results show that the gold fineness decreases as distance is gained from the battery; also that the proportion of bullion to the amalgam also decreases in

the same direction. The tool used to remove the scale from the plate was a common putty-knife, having the blade cut off to about one-half its length, and dressed with a file to have a slight bevel. This edge was further dressed on an oil-stone, so that it would loosen the scale without "scraping" so as to expose the copper. The plate was left in excellent condition, and, after a few hours, was perfectly normal.

After crushing 200 tons, the guide-blocks, set in the top of the mortar, were considerably worn, especially at the front of the stem. Strips of old rubber belting were inserted in these openings to keep the stems in place, and were found to work admirably, obviating the necessity of putting in new guide-blocks. These strips had to be renewed about every two weeks.

A number of tests were made to determine the percentage of possible amalgamation, which was found to range from 48 to 60 per cent., averaging 50 per cent. In actual practice, the percentage ranged from 40 to 55 per cent., averaging 48 per cent. It is safe to say, therefore, that the amalgamation was practically complete.

It would seem that the field of usefulness of the steam-stamp in connection with amalgamation is bound to increase in proportion as it becomes better known.

The Ore-Shoots of Cripple Creek.

BY EDWARD SKEWES, CRIPPLE CREEK, COLO.

(Colorado Meeting, September, 1896.)

Introduction.

In view of the importance of the Cripple Creek district, the large amount of the publications concerning it, and the circumstance that many members of the Institute reside in it, or are familiar with it through repeated visits, it is surprising that our *Transactions* should have remained so long without any contributions on the subject. The same observations may be made with regard to the Witwatersrand, in South Africa, the other great productive gold-field of recent development. And I suppose the explanation is the same in both cases, namely, that the members of the Institute personally acquainted with the

economic geology of these districts have been so intensely and continuously employed in their actual development that they could not find time to prepare, for the information of professional students, general descriptions of the geology and ore-deposits of the regions concerned. Fortunately, in the case of Cripple Creek, the deficiency has been made up by a report of the United States Geological Survey, by Messrs. Whitman Cross and R. A. F. Penrose, Jr.,* which reflects the highest credit, not only on the authors for their skilful and thorough work, but also on the Director of the Survey, for the early inauguration and vigorous prosecution of such an investigation, and the prompt publication of its results. Without disparagement of the previous work of the United States Geological Survey in the Rocky Mountain region, it may be frankly said that many of the important monographs of the Survey in economic geology have appeared after so much delay in preparation and in printing as to constitute rather contributions to history and to general science than immediate aids to the intelligent exploration and exploitation of the districts described. No doubt, scientific accuracy and permanent value are promoted by such delay (provided it is not wholly, as has been sometimes the case, on the part of the government printer), and no doubt, also, the publication of earlier reports, based on less extensive underground data, should not take the place of later and more mature utterances; nevertheless, the prompt appearance of scientific official reports from competent and disinterested sources is a boon to prospectors and mining engineers, for which they cannot be too grateful. Though such general descriptions may require subsequent enlargement or correction in details, they serve an immensely important purpose in simplifying and clarifying popular views, unifying popular nomenclature, and guiding explorations. Concerning the work of Messrs. Cross and Penrose, it is only fair to say that it combines with this quality of practical usefulness so high a degree of scientific accuracy that the question, whether its general theoretical conclusions will be impaired hereafter can only be answered, at this time, in the negative.

As an introduction to the observations presented in this pa-

* *U. S. Geol. Sur.*, 16th Ann. Rep., Part II., pp. 13–209.

per concerning a limited part of the general subject, the following statements are quoted or condensed from the report mentioned:

Mr. Cross describes the Cripple Creek district as situated in a group of hills, mostly rounded, 7 to 12 miles southwest of Pike's Peak, and 9000 to 10,000 feet above sea-level. These hills consist almost entirely of volcanic rocks, and indicate the successive phenomena of a true volcano; the principal material being breccia or tuff, resulting from a succession of explosive outbursts, while the igneous magmas erupted were, first, andesite; then phonolite (in numerous eruptions, alternating with explosive outbursts); and finally, a number of highly basic eruptives in small dikes. Gaseous exhalations have played a part at certain periods, and "in the final stage of the volcanic cycle, hot waters circulated through the more or less permeable mass, and decomposed the rocks to a great extent. The formation through which the Cripple Creek volcano burst, and upon which its superficial materials lie, is the granite-gneiss complex of the Colorado range." Since the close of volcanic activity, "the granitic plateau and the volcanic accumulations upon it have been greatly modified by erosion."

Mr. Penrose summarizes the geological conditions as follows:*

"The gold of the Cripple Creek district is found both in vein deposits and in placer deposits derived from the decay and erosion of the veins and country-rock. Though the placers have produced considerable quantities of gold, the veins are far more important and supply most of the gold of the district. They generally occur in fissures in the country-rock, which usually represent slight faulting, while more rarely they occur in other positions. The veins intersect all rocks in their course and have been formed mostly by a replacement along the fissures and not by the filling of open gaps.

"The district is essentially an elevated area of Tertiary volcanic breccia cut by numerous massive eruptive rocks in the form of irregular bodies and dikes, and surrounded on all sides by granite. Besides the intrusive bodies of eruptive rocks, the breccia area also contains massive eruptive rocks which were formed before the breccia area and which now represent included bodies. Bodies of granitic rocks are also included in a similar manner. The region of eruptive rocks is about 3 miles long in a north-and-south direction and about 2½ miles wide in an east-and-west direction, comprising an area of about 7 square miles.

"From later Cretaceous well down into Tertiary times, eruptive outbreaks were active throughout the country now occupied by the Rocky Mountains, and the vast quantities of lavas, breccias, tuffs, and eruptive materials of other kinds found in this region from Canada to Mexico were poured out largely during this period. The area now occupied by the mining district of Cripple Creek is in part

* *Op. cit.*, pp. 137, 138.

on the site of a volcanic vent, or possibly of more than one vent, formed in Tertiary times. The region originally consisted entirely of granitic rocks, such as now compose most of the Pike's Peak range, but during the period of volcanic activity a vent or series of vents was formed, and was the scene of a series of explosive eruptions which ejected large quantities of eruptive rocks in a fragmental condition, thus forming the volcanic breccia which now comprises the larger part of the district and which includes bodies of the original eruptive rocks from which it was derived. Much of this breccia was thrown over the immediately surrounding granite, and at the end of the time of eruption the vent or neck itself was filled, choked up as it were, with the same breccia, so that the present area of volcanic materials is underlain partly by granite and partly by the filled-up vent.

"After and probably during the outbreak of breccia, eruptive masses of andesite, phonolite and other rocks were intruded into the breccia area in the form of large bodies and of dikes. Some of these rocks overflowed a little on the surface, covering limited areas; but some of them, as seen now, show no overflow, though such may have existed once and may since have been eroded. The bodies of massive eruptive rocks are not confined to the breccia area alone, but also occur in the surrounding granite area, as seen on Mount Pisgah, Beacon hill, Grouse mountain, Little Pisgah mountain and elsewhere, while the whole of the immediately surrounding granite area is much cut up by dikes in the same manner as the breccia.

"The dikes intersect all other rocks in the district, and pass indiscriminately from granite to breccia and from breccia to granite. They were the last signs of eruptive activity, and their appearance is evidence of the epoch of fissuring which lasted long after the dike action had ceased. They are of various compositions, and from a few inches to many feet in width, generally, however, averaging from 1 to 10 feet. The fact that they intersect each other is evidence that they were formed at intervals during a certain period in the geological history of the district."

Concerning the occurrence of the ores, the following observations are compiled from Mr. Penrose's report:

1. The vein-fissures do not seem, as a general rule, to have been the earliest ones. The fissures filled by the dikes seem to have been generally, though not always, earlier.

2. The general course of the vein-fissures, like that of the dikes, varies from northeast to northwest, and in many cases is nearly north and south. There are, however, a few prominent exceptions.

3. The vein-fissures were not open gaps, but almost closed lines of fractures, and the vein-filling is largely due to a replacement of the country-rock.

4. These fissures were lines of movement, though the faulting was rarely extensive. Parallel cracks, etc., constituting fissured zones, are frequent.

5. The vein-fissures traverse both the brecciated and the solid rocks, including the country granite, and their nature is

much affected by the character of the rocks they intersect—the fissures being much better defined in granite massive eruptives or hard breccia than in more plastic rocks, like soft, tufaceous breccia.

6. The ore-bearing vein-materials are bodies of secondary minerals filling the fissures. They occur in all the rocks—granite, massive eruptives and breccia—and have been followed many hundred feet in depth, without showing signs of exhaustion. "The ore generally appears to be simply the country-rock, containing greater or less quantities of secondary minerals, which decrease in amount with distance from the fissure along which they were deposited." Many fissures are barren.

7. It is frequently found that the ore-bearing veins follow the dikes for greater or less distances. Mr. Penrose holds, however, that the veins are seldom true contact-formations between the dikes and the country-rock. In his opinion, the vein-fissures, being generally of later formation than the dikes, have been simply affected in their course (and perhaps also in the deposition of their mineral contents) by the pre-existing dike-fissures which they have encountered. The typical relation between the two fissure-systems, according to his view, is that of a more or less oblique intersection, with a deflection of the (later) vein-fissure, due to the line of structural weakness in the rock produced by the previous dike-fissure. The latter may have furnished also water-channels, shrinkage-cracks, etc., favoring ore-deposition.

8. The gold occurs in the ores as telluride of gold, free gold (derived from the decomposition of the telluride), and possibly in auriferous pyrite—the last being the least important occurrence.

9. The gold is not uniformly distributed throughout the vein-fissures, but is relatively concentrated in ore-shoots. Mr. Penrose says (page 162 of Report):

"The Cripple Creek shoots are of varying shapes and richness, and they trend in various directions in the fissures, though possibly in many of them a general southern pitch down and along the fissure is more common than any other. This is most often noticeable where the course of the shoot is guided by certain transverse fissures, to be mentioned later. Elsewhere the shoots may dip vertically, and more rarely a little to the north. They vary from one to several hundred feet along the fissures, and from a few inches to several feet in thickness. In some places they have a well-defined columnar shape, in others they have forms of no

regularity whatever. In some places they outcrop at the surface, in others their apex is many feet below the surface; in some places they are very continuous in depth and extend as deep as they have yet been followed, in others they come to an end at a comparatively shallow depth, though occasionally these shallow shoots are replaced by others at a greater depth."

By way of further preliminary explanation, the following rough geographical statement may be of use:

The observations recorded in this paper cover an area of about 12 square miles, and for convenience, the reader may imagine a territory 3½ miles square, having the town of Cripple Creek in the northwest, and the town of Victor in the south-east quarter, the distance by air-line between the two being about 3 miles. At about the center of this square would be Raven hill; a mile N.W. of which is Gold hill; about the same distance N.E., Bull hill; a mile and a half S. of Bull hill, Battle mountain (in the S.E. quarter of the square), and a mile W. of Battle mountain, Beacon hill (in the S.W. quarter)—these distances being roughly estimated by air-line. There are many other hills in the district, but these are here enumerated because they are the localities of the mines hereinafter mentioned. All of them (except Beacon hill) consist at the surface of andesitic breccia and tuff, with underlying granite and dikes of phonolite, etc. In Raven hill the basaltic dikes are most numerous, and the veins are most closely associated with them. In Gold hill the characteristic breccia of the region predominates, and in this hill is the S.E. termination of a schistose dike, the N.W. extremity of which is in Mineral hill, more than a mile away (on the northern end of the square). In Bull hill the massive eruptives occur, with typical phonolite, and the rocks are extensively fissured. Battle mountain is almost entirely surrounded by granite and abounds in andesite. Beacon hill is characteristically phonolite.

The main purpose of this paper is to put on record some facts concerning the occurrence, and more particularly the pitch, of the ore-shoots in the localities named.

Raven Hill.

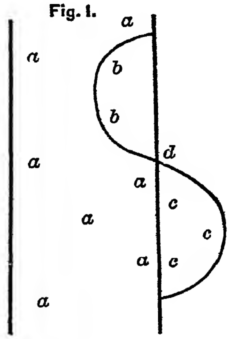
Doctor.—The course of the Doctor vein is N.E. and S.W.,* with a dip to the N.W. at the surface of 38°, and at greater

* This is one of the few claims in the camp the vein of which runs through the location parallel with the side-lines.

depths of 60°. The ore-shoots pitch with the dip of the vein to the N.W. In June, 1895, there was seen in this claim the best surface-showing ever witnessed in the camp. The characteristic cubes of gold varying in width from $\frac{1}{8}$ to $\frac{1}{4}$, and in length from $\frac{1}{2}$ to $\frac{3}{4}$ inch, were abundant through an uncovered area 12 inches wide and 15 feet long.

There are three well-defined ore-shoots on this vein, all having the same pitch.

Fig. 1.



Plan of Shoot in Doctor Vein.

None of the shoots show improvement in values with depth; in fact, the richest ores were mined within 60 feet of the surface.

The North Star vein, on the Doctor claim, has a N.-S. course and dips W. The ore-shoot pitches north. At a depth of 360 feet the shoot yields from a 3-foot vein the best ore it ever has shown.

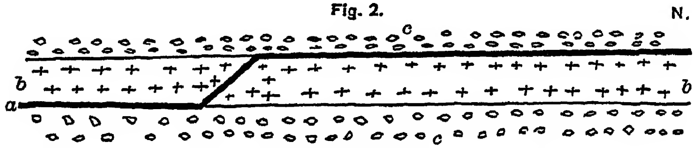
A peculiar feature appears in the main shoot of the Doctor vein, the like of which I do not know elsewhere. Fig. 1, a diagram-plan, will give a rough idea of its shape, not unlike the letter S. The portion *b*, *b*, although within the boundaries of the vein, which has well-defined walls, is waste rock. Its physical characters are similar to those of other parts of the shoot, but without value; whereas *c*, *c*, beyond the limits of the vein, carries value. This letter S has occurred on the hanging-wall twice. The volumes of *b* and *c* were approximately equal. The pay-streak is richer on the hanging-wall, and only twice has it changed or crossed from the hanging- to the foot-wall.

Chief.—The course of this vein is N. and S.; the dip may be said to be vertical, as, at a depth of 280 feet, the vein is less

than 10 feet from the shaft. The pitch of the ore-shoot is to the north. Dr. Penrose says: "The ore occurs in more or less close association with two intersecting dikes in the breccia, one striking approximately north and south, and the other a little north of west and south of east." The ore-shoot pitches to the north-and-south dike.

Mr. Cone, one of the owners, to whom I am indebted also for information respecting the Doctor, told me that, when the shaft reached 210 feet, a nearly square chimney of mineralized rock 10 inches square, of an average value of $75 per ton, made its appearance, and continued in the shaft for 16 feet. At the depth of 226 feet the chimney broke off abruptly. It had no

Fig. 2.

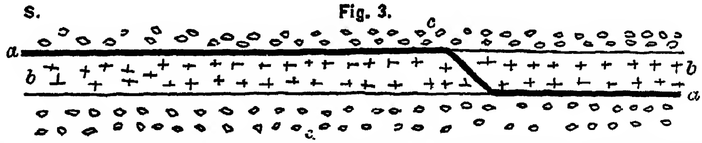


Elkton Vein, Horizontal Section above 1st Level. *a*, pay-streak; *b*, vein; *c*, breccia, country rock.

connection with the vein, and, when first struck, was supposed to be the apex of a new ore-shoot.

Elkton.—This claim and the Walter, mentioned below, are now both owned by the Elkton Consolidated M. and M. Co. In October, 1893, I examined the shaft, then about 25 feet deep. At that depth the vein was fully 6 feet wide, consisting of square blocks of rocks, resembling a poorly constructed massive wall without any mortar. The joints were from 6 to 8 inches apart, and the rock-masses varied from 2 to 3 feet in diameter

Fig. 3.



Elkton Vein, Horizontal Section on 1st Level. *a*, pay-streak; *b*, vein; *c*, breccia. country rock.

and were best mined with crowbars. The first shipments, without any sorting, yielded 5 ounces of gold per ton. The richest part of the vein was a sand-quartz, stained with fluorite, in which there was considerable free gold.

The course of the vein is N. 2° W. and the dip 85° east. The Elkton shaft is vertical. At 300 feet and at 400 feet the vein was intersected by cross-cuts, 12 and 18 feet long, respectively. The pitch of the ore-shoot is 75° to the north.

The walls at the third level are almost perfect for 400 feet, especially the east wall.

Fig. 2 is a diagram of the horizontal section of the vein, as seen in August, 1894, 25 feet above the first or 100-foot level (which has since been stoped) and 40 feet S. of the discovery-shaft. The pay-streak crossed from the E. to the W. wall as shown. I sampled the pay-streak where it crossed the dike, and, for the 5 feet across the dike, it assayed $80.40 per ton. The main part of the dike carried only a few dollars' value, and was left on the stulls.

Fig. 3 is a horizontal section at the first level, 60 feet north from the main shaft. Here the pay-streak crossed from the W. to the E. wall. My sample from the portion within the dike, however, carried no values, although the pay-streak when on the walls was of the regular values. The vein, after leaving either side of the dike, continues for a distance of about 20 feet as a sheeted iron-stained rock, 1 inch wide, which gradually tapers out entirely.

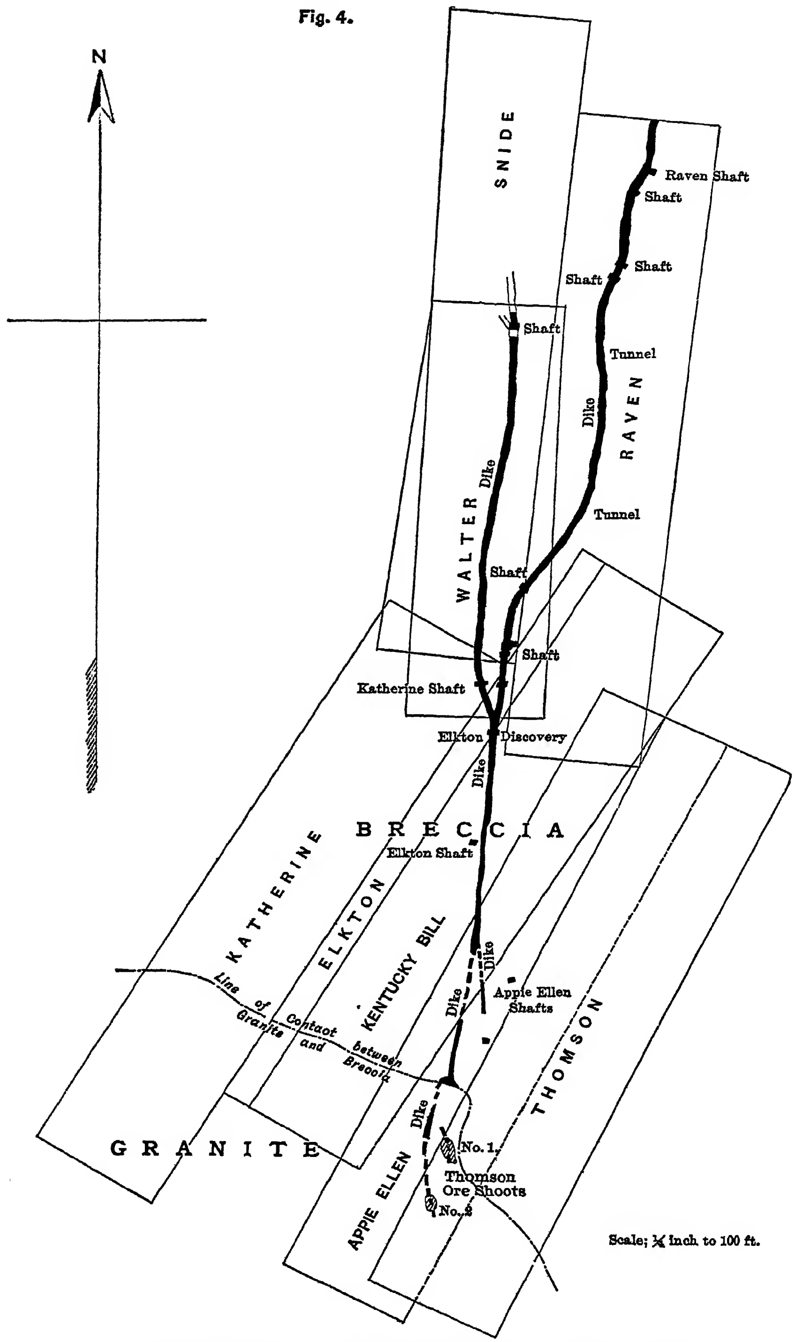
The course of the dike, in and along which the pay-streak is found, is the same as that of the vein, the pay-streak being really an altered part of the dike, the rock being partially or wholly replaced by quartz, iron oxide, fluorite, etc. All the dikes on Raven hill are enclosed in the breccia.

The depth of the superficial alteration of the rock was approximately 170 feet. The ore at that depth was largely a porous quartz, stained throughout with fluorite. The vein was not at all sheeted.

Walter.—This claim is 800 feet in length. The dike after leaving the Katherine claim, passes through the Walter claim, in which it has been opened for 740 feet in length (see Fig. 4). The Raven dike (which is the Elkton dike) and the Katherine or Walter dike unite between the Katherine shaft and the Elkton discovery. The course of the Raven dike is N. 5° E., that of the Walter dike is N. 2° W., and from the point of meeting of the dikes the course is about due south.

In this hill there is a decided tendency for the vein and dikes

Fig. 4.



DIAGRAM, SHOWING COURSE OF RAVEN AND KATHERINE DIKES.

to split as they travel north, or up the hill. In the southern part of the hill there is only one well-defined vein or dike, whereas northward the one dike becomes two dikes on the north end of the Elkton claim; one part (the Raven dike) maintaining its normal course, the other part taking a W. of N. course. Near the breast of the Elkton No. 1 level, 740 feet to the north of the Walter line, the latter dike is split again, one branch (driven on for 25 feet) going N. 8° W., the other part taking the usual direction; and still further north, several other veins, having the same general strike and dip and composed of much the same material, have been found. Similar conditions exist on Battle mountain, where on the south end of the south slope, the Independence and the Strong are the only well-known veins; while on the north end of the south slope there are a dozen veins. On the south end of the south slope of Gold hill, the only well-known vein is the Anaconda, while on the north end of the south slope, and on the apex, there are many well-known veins which are producing ore. The same conditions exist on Bull hill. Globe hill has nothing in common with the other hills of the camp, and since 1892 has been "the black sheep."

Such a general feature indicates that vein-formation in this camp was not unlike the growth of a tree—the trunk first, the branches next. In the *Engineering and Mining Journal* of February 16, 1895, page 151, I said:

"I find in every place examined that in the contact or intersection of a vein with the phonolite dike, the dike has been the intersecting medium, and there is scarcely a case where it has in the least faulted a vein. The eruption of the dikes, I am inclined to believe, was not violent, as the enclosing breccia-rock is not altered or metamorphosed, or, if so, it is not observable."

The contour of the surface indicates nothing of the explosive type of eruption.

As may be seen by the accompanying sketch, Fig. 5, of the Walter, one of the Katherine ore-shoots pitches into the Walter at an angle of 45°. The same shoot may be plainly seen in Elkton No. 2 level in the Walter ground, Elkton No. 3 level not being sufficiently advanced north to encounter it. This shoot is 30 feet in length, and left the Katherine ground 60 feet above the Elkton No. 1 level. After that ore-shoot has been passed in the level's northward course, a barren piece of

ground 80 feet in length was encountered, which being passed, the main ore-shoot comes in from the south with a pitch of 70° toward the north, and continues north to the present breast 600 feet.

This long shoot is not rich, although there are rich shoots and pockets in it. The average width has been fully 4 feet, 25 per cent. of which samples by car-load lots $2\frac{1}{2}$ ounces of gold per ton. The pay-streak in this long shoot is not at all regular, but consists of a greenish quartz, often found in the center of the vein, and always associated with free gold. The walls are much broken up, and where the rich shoot is, as shown in Fig. 5, the walls are very much softer, as is also the vein. In the barren ground reaching from Walter 2 to Elkton No. 1 level the vein was very much harder, and the walls consisted of large blocks. Soft ground in the Walter may be regarded as a favorable sign.

It may be added that north of the rich ore-shoot (which sampled over 20 ounces of gold per ton for 12 inches wide and 30 feet in length) the pay-streak was always on the east side; whereas south the pay-streak was on the east, on the west, and in the middle.

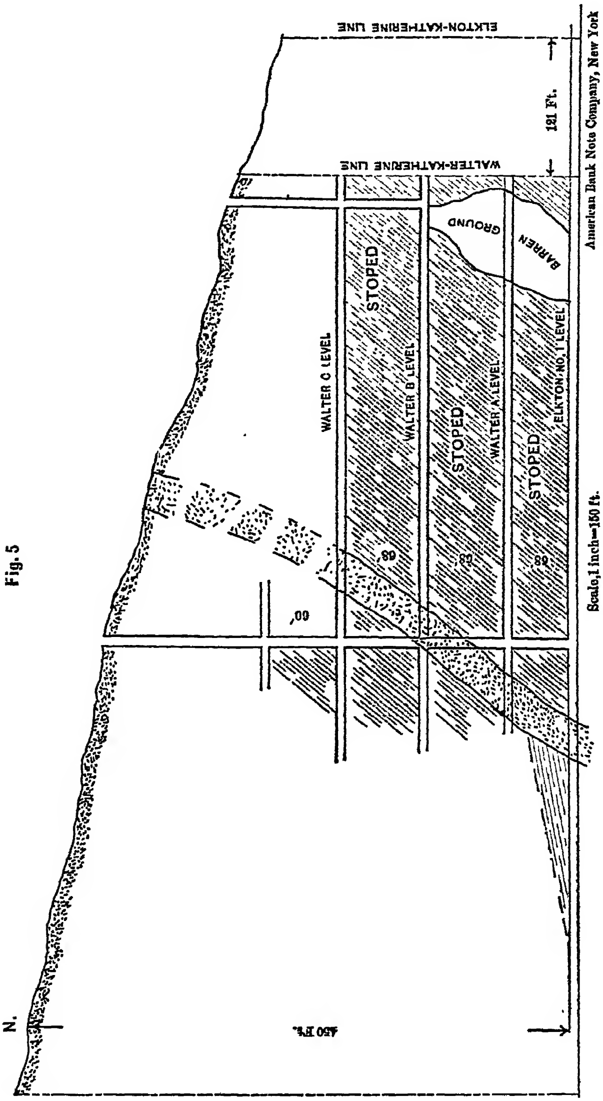
As stated above, the Katherine ore-shoot was found on the Elkton 2d level in the Walter ground. Where it was found, the ground was comparatively soft, and the east wall of the vein showed a variation of five degrees to the east; but it soon regained the usual course, N. 2° W. The shoot is approximately of the same length, 30 feet.

At the Elkton 3d level, in the Walter ground, about 100 feet north of the south boundary-line, an entirely new ore-shoot or pocket was found, 25 feet in length, 6 feet wide and of the form as seen in the roof of the level shown in Fig. 6. This pocket was enclosed in the regular breccia. By simply driving through the ore-shoot it was impossible to learn the pitch. The ore was strictly a telluride, scattered through small seams. The vein-material was full of small vugs or crevices; otherwise it was the ordinary breccia mineralized. From this pocket were taken three car-loads of ore, which yielded three ounces of gold per ton without sorting.

This shoot, encountered at a depth of 520 feet, is another refutation of the theory which has been advanced respecting

Cripple Creek ore-shoots, namely, that all the ore-shoots come to the surface. There are, in fact, many shoots whose apexes

Fig. 5



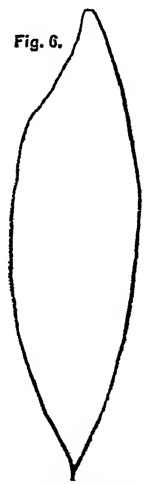
Longitudinal Section of the Walter Mine.

are found at from 80 to 500 feet below the surface. The spread of such a theory has been the cause of much unwise prospect-

ing. The present systems of prospecting are: 1. To trench or "costean" the whole surface of the ground; 2. To sink a vertical shaft 200 feet deep and cross-cut at that depth to the boundaries.

Thomson.—This claim lies south of the Elkton group, which includes the Kentucky Bill, Elkton and Walter. Fig. 4 shows the course of the dike in crossing the Appie Ellen and entering the Thomson claim; it also shows the contact-line between the granite to the south and the breccia country-rock

Fig. 6.



Form of Ore-Shoot shown in Roof of Elkton 3d Level, in Walter Mine.

northward. The ore-shoots of the Thomson are in proximity to the course of the dike. There are on this claim two chimneys, entirely in the granite. No. 1 measures about 70 feet from N.W. to S.E., by 15 feet maximum width, and has been followed from the surface to the depth of 203 feet. The main mass of this ore is low-grade. The dike shows a few isolated fragments very different and distinct from the mass of the dike-rock. Shoot No. 2, another pipe or chimney, located 120 feet south of No. 1, has been followed only 60 feet from the surface. Its average size is 12 by 20 feet in cross-section, and it contains a larger proportion of pay-ore than No. 1. The dike, so far as disclosed, is not immediately in contact with the ore-body. The maximum width of the dike is only 6 inches, and so far as developed, it appears just as fragmentary as in shoot No. 1.

There is about 2 feet of barren ground between the dike and the ore. In fact, the basalt dike, as it enters the granite, seems to split into several stringers, and thus far has not been found south of the Thomson ground. There has been no ore found along the line of the granite-contact in any of the claims shown in Fig. 4.

Raven.—The length of the ore-shoot in this mine (see Fig. 7) varies from 100 to 240 feet, and the pitch is about 65° to the south. The pay-streak, like others in Raven hill, changes from the foot- to the hanging-wall and *vice versa* without any increase or decrease in value. It is difficult to tell, except by assays, where the value in the vein ceases. The gouge and the walls still maintain their course and character, as also do the rocks when macroscopically examined.

This vein has been but little oxidized, although often free gold and telluride can be seen in the rock from the greatest depth (460 feet) yet reached. The vein was not oxidized below 100 feet (whereas the Walter, adjoining, is completely oxidized at the depth of 450 feet); but the tellurides are undergoing decomposition at the present time, with free gold in the form of the original telluride crystals as a result.

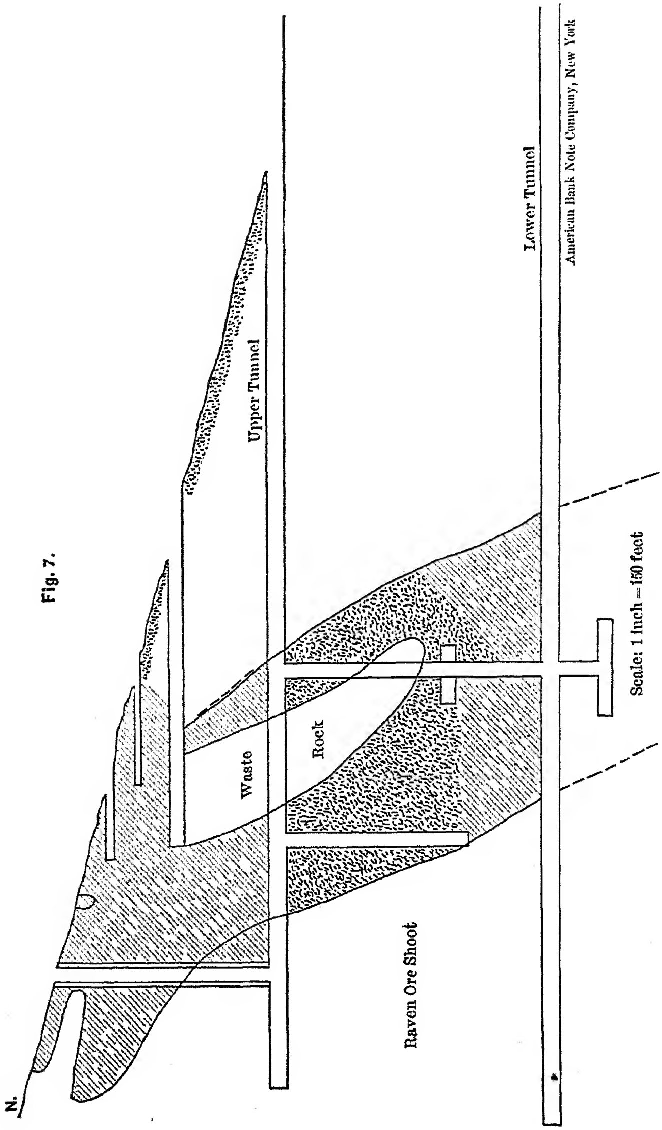
The course of the Raven vein is in the main N. 2° E., and its dip 75° to the west of north. The walls are well defined, and the vein has an average width of 4 feet, rarely pinching below that point, but occasionally swelling to 10, 12 and in one place to 16 feet wide.

This is the only vein on this hill known to me in which slickensides or *striæ* have been found. The slickensides present a bright metallic luster, often called native silver, showing considerable friction in upward or downward movement. The *striæ* are much like corrugated iron on a small scale, the indentations being about ¼ of an inch deep. The rock is the ordinary breccia.

This vein differs from most of the veins of the hill in the following particulars: it is relatively unoxidized; changes are apparently taking place in the vein daily; while the other veins follow the line of the dike, this dike follows the vein for 600 feet; it is the only vein which has an ore-shoot pitching south; it is the only one which exhibits slickensides or grooves; the character of the ore is different, in that it contains less quartz or fluorite than usual.

Moose.—This vein has a course in the main of N. 12° W. and a dip of 80° E. by N. There are two ore-shoots, one north

Fig. 7.



Longitudinal Section of Raven Mine.

and one south of the shaft. The north shoot pitches north at an angle of 82°, whilst the south shoot pitches north about 68°.

The shaft has been sunk 650 feet, and during the last few feet of sinking a little ore was broken, indicative, perhaps, of the apex of another ore-shoot. The influx of water stopped the work. This is one of the few mines of the camp that yield silver largely. Less than 12 months ago, two car-loads were taken from the south ore-shoot, which yielded 54 ounces gold and 114 ounces silver per ton; the silver being in part native. The average of the silver-value, however, is less than $5 per ton.

Gold Hill.

Moon-Anchor.—This vein varies in course from N. 26° E. in the southern portion to N. 60° E. in the northern, and dips 75° S.E. The ore-shoot is about 70 feet in length. The width of the pay-streak is 3 feet. It lies on the hanging-wall, and, unlike the Raven hill pay-streaks, it has not changed to the foot-wall, and there has been no change whatever in the course or dip of the vein.

This ore-shoot was formerly supposed to dip south. The reason is plain. At and near the surface the ore-shoot was funnel-shaped, and most of the work was done, and the principal part of the ore was extracted, from the north side.

The regular vein, in the proximity of the ore-shoot, shows slickensides on the hanging-wall.

Occasionally the ore-shoot yields mineral from foot- to hanging-wall. This is one of the few mines of the camp where water-level, or the line of permanent saturation, has been reached. The influence on the ore-shoot or the vein was very feeble.

Geneva.—This claim attracted considerable attention during the summer of 1895. The ore-shoot was first encountered at from 70 to 80 feet depth by the sinking of a vertical shaft. From that depth the vein inclines to the east, or into the hill, at an angle of 70°, until 160 feet is reached, the course being north and south. From 160 feet to 450 feet, the shaft was sunk on the vein, which is vertical. The shoot of ore lasted from 80 to 160 feet, and the length was about the same as the depth, namely, from 80 to 85 feet. The average width stoped was 12 feet; in one place 20 feet.

The ore was largely a phonolite, the seams and faces being often covered with a tin-white telluride, such as is rarely found

in the hill. The rock in which the telluride was found is a pale or white, fine-grained phonolite, easily fractured, and clearly distinguished from the barren part of the vein or dike in that the unproductive rock is of a dull, heavy black color. The pitch and course of the ore-shoot are difficult to determine, for the reason that it is worked out, or nearly so. The workings indicate a north pitch at an angle of from 60° to 65°. At the first level (80 feet) the stopes are 20 feet south of the shaft, whereas at 160 feet the stopes are all north.

The vein as seen below the pocket was 4 feet wide. At the 400-foot level the vein was intersected by an east-and-west vein 100 feet north of the shaft, having a dip of 82° to the south. The influence of the cross-vein was imperceptible; the same remark does not apply to an east-and-west vein found in the ore-shoot or pocket.

Anchoria-Leland.—The course of this vein is N. 20° E. in the main, but varies several degrees for short distances. The dip frequently changes, sometimes the east wall is the foot-wall, and sometimes the hanging-wall. The ore-shoot is over 500 feet in length at the first or 278-foot level, and pitches north. The walls are not to be seen opposite each other; so the vein cannot be said to be "well defined." Where there is a wall the parting is often indistinct; there is no gouge. Gold hill, more than any other hill in the camp, is characterized by the absence of fluccan or gouge. The gradation between vein-matter and country-rock is often imperceptible, and frequent assaying is indispensable to the determination of values. The absence of walls was one, if not the principal, cause why this hill was so much longer than the others in being developed. Prospectors were prejudiced, because where a wall was found there was no clean-cut parting; the vein was "frozen" to the country.

The alteration from a barren andesitic breccia to one containing value in the neighborhood of the vein, obliterating every trace of walls, is due solely to the mineral solutions; the country-rock in such places affording no resistance.

In this vein or ore-shoot there is no evidence of sheeting. We have simply a mass of eruptive rock varying from 3 to 17 feet in width, and irregularly impregnated with materials carrying variable gold-values. The pay-streak at the 278-foot

level is, as a rule, very hard, and consists of a very dark, almost black, glassy quartz, accompanied by small vugs, which are partially oxidized. Near the shaft the pay-streak is on the east, and as the level proceeds north the pay-streak is on the west wall. The change did not affect its value. Unlike those of the Raven hill mines the pay-streak here cannot be traced where it crosses the vein.

At the second or 344-foot level the south level only is being driven, and the pay-streak is apparently lengthening, the dip here being 88° S.E. This is one of the most interesting cases in the camp. For the first 100 feet in depth, the vein dipping to the N.W., the mine was but partially self-supporting; but as soon as the vein straightened itself, and subsequently dipped to the S.E., the profitable stopes became 14 feet wide. I noticed no cross-veins, no seams, nor any alteration in any way in the vein, but simply the change of dip.

The east vein was first discovered at a depth of 100 feet, and 125 feet north of the Dickerman or main shaft. It has not yet been found at the surface. The course is S. 65° E. No work has been done on this vein below the 278-foot level. At this place the vein is 4 feet wide, with a good north wall, vertical. The pay-streak is well defined and well oxidized, containing free gold. Sixty feet above the 278-foot level and 50 feet east have been recently found some of the richest gold and telluride specimens ever encountered in the camp. The pitch of the rich seam was 25° to the east, and it has been opened in length for 60 feet. It was from 1 to 2 inches wide, located in the middle of the vein, about 60 per cent. of the pay-streak being oxidized. I examined this part very carefully, but could find nothing to account for the increased richness, save that the vein changed its course 7° to the north. The rich seam, as well as the vein, was vertical. The south wall was badly fractured.

Last July demonstrated that in this mine the grade of ore improved with depth; the average of over 5000 tons previously produced above the 200-foot level was $43 per ton, whereas the average of July was $125 per ton, all mined below the 260-foot level.

C. O. D.—The shaft on this claim has been sunk 350 feet. The strike of the vein is about N. 15° E., the dip being 80° east-by-south. The ore-shoot may be said to be one of the best-defined in the camp. It is 125 feet long, with a pitch to the

south (from 7 feet to the bottom of shaft) of 50°. The walls of the veins are well grooved and slickensided. The largest pump and the greatest volume of water to handle in the camp are here. The pump has been idle for three months, and the water does not rise to the 300-foot level.

Abe Lincoln and Arcadia.—The course of this vein is S. 75° W. The vein is vertical. The ore-shoot has been disclosed for 150 feet on the two claims. The width of the vein, as opened for 275 feet, varies from 1½ to 8 feet. The vein-matter is a trachytic phonolite, with granite on both sides. The granite on the south side, or Gold hill, is about 75 feet wide, in which other small veins have been found on the Arcadia ground. The ore-body is characterized by a seam of talc which changed to crystallized quartz occasionally—rather unusual in this camp. The ore is found on both sides of the seam in equal proportions. The seam is very irregular in its course, sometimes on one side of the dike and sometimes on the opposite side, and sometimes passing into the granite entirely. The ore-shoot, so far as disclosed, pitches east.

The granite is entirely unaltered here, perhaps more completely so than anywhere in the district. Mica and feldspar are the two principal constituents.

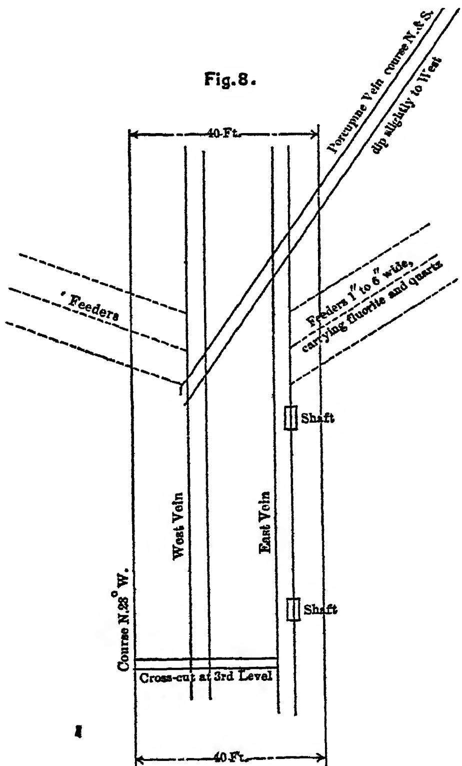
City View.—The strike of this vein is N. 20° E., and the dip from 70° to 85° N.W. As yet there is no well-defined ore-shoot, although shipments are regularly made from small seams, carrying value of from $500 to $1000 per ton, and driving further north in these seams will doubtless encounter the regular vein. What has interested me most here is a "slip" at the 185-foot level which has cut the vein entirely out. This level was extended 142 feet 4 inches on the vein N. of shaft, the course of the vein and distance from the breast to the shaft being S. 24° W., 26 feet 4 inches; S. 45° W., 65 feet; S. 29° W., 25 feet; and S. 27° W., 26 feet.

At a point 116 feet 4 inches from the shaft a well-defined vertical wall, having a strike of N. 30° W. faulted the vein, which has not yet been found to the N. beyond it. For nearly 45 feet the vein in the level was profitable, three car-loads of ore were shipped, and good ore was found against the slip. There was no evidence of movement on slip or vein. The breccia near the seams is much silicified for 8 inches.

Bull Hill.

Orpha May.—This vein is 40 feet wide, and courses N. 28° W. It includes two other veins, each 3 feet wide (see Fig. 8) known as the "East vein" and "West vein," in which six ore-shoots have been found. Between these two veins is a distance

Fig. 8.



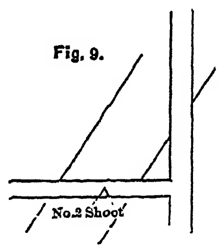
Plan-Diagram illustrating Vein-Relations at Orpha May Mine.

of 15 feet, which does not show parallel zones of fissuring, but rather a mixture of cross and parallel fissuring, seams of quartz running in every direction, and carrying values, but too narrow to be worked, yet enriching the entire 15 feet, so as to make it assay about $8 per ton. The East vein and the West vein are worked at the 4th, 5th, 6th and 7th levels. The former dips more to the S.W. than the latter, which has the same course as the vein in which it is enclosed, and dips to 80° S.W.

The six ore-shoots opened have an average length of 50 feet each on the 900 lineal level feet of developed vein. Five of them pitch south, the angle varying from 35° to 80°. The north ore-shoot pitches north, but there are no developments below the second level in that direction.

The pay-streaks are generally on the foot, but occasionally jump to the hanging-wall; no matter where the pay-streaks are, they carry equal value. A variation of a few degrees in the course of the vein does not enrich or impoverish it; whereas a difference in the dip of the vein, say from 2° to 10° in the

Fig. 9.



Top of an Ore-Shoot within an Ore-Shoot, Orpha May Mine.

foot-wall, irrespective whether that dip is to the east or west, does give additional value, and the rich ore often comes from the pockets found in the alteration of dip. Sometimes only a few pounds, and at other times a few tons, are taken out at such a point. In this respect the mine resembles the Anchoria-Leland. These changes in the dip of the vein bear an analogy in part to bends in the bed of placer mines serving as receptacles for the richer minerals.

By reference to Fig. 8 it will be seen that the Porcupine, a N. and S. vein, intersects both the "East vein" and the "West vein." The ground is very much broken at the junction. Early in 1895 I made the sketch shown in Fig. 9, for the reason that it was the first time I had seen an ore-shoot within an ore-shoot. The apex of the second shoot reached 4 feet above the level, and was like the figure $\wedge$ in shape, being 3 feet wide at the bottom of the level. This shoot was very rich. It was encountered as shown in Fig. 9, on the 80-foot level of the Pike's Peak.

Pike's Peak.—This vein has a course of N. 12° W., and stands almost vertical. The location is at right angles to the vein, and

for its full width of 300 feet yields mineral from what may be said to be one shoot. The vein is small, varying from 2 to 12 inches to the depth of 336 feet. This mine, like the Orpha May, is still in the oxidized zone.

In June, 1893, four miners took a lease on this claim, and after sinking a disused 12-foot shaft some 3 feet further, encountered on the west side a mud-seam varying in width from ½ to 2 inches. The material, which was purely a surface-mud, as far as could be distinguished by physical characters, was very rich in both angular and rounded forms of gold, the pieces of which were of large size, even for this camp. Thirty pounds of gold were "panned out" in the course of a few weeks. The lowest degree of fineness given by the Denver mint was 992.5, the highest 999.2. The residue from the pan assayed from $2000 to $6000 per ton. There were no defined walls at that time; the mud followed the crevices in the rock, and so careful were the miners that they scraped the mud from the rocks with knives.

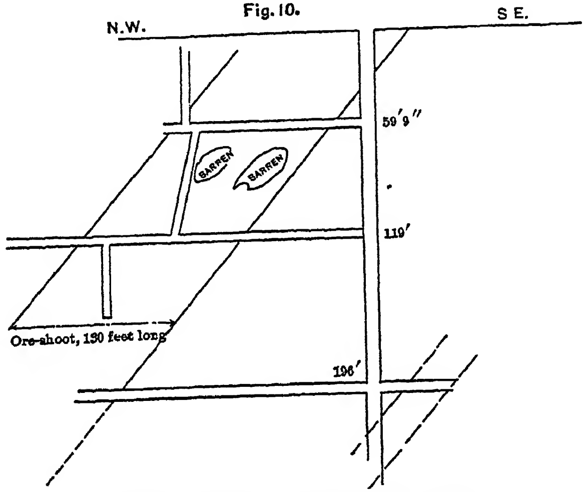
Pharmacist.—This vein has been worked in the Zenobia and the Burns, and connections have been made with these mines. The course of the vein varies from N. 50° E. to N. 65° E., and it dips west of north from 65° to 75°. The shoot of ore or, perhaps more accurately, the two pockets, discovered in 1892, which enabled the company to pay $84,000 in dividends in a few months, were, to judge from a cursory examination of the stopes, V-like in shape; and near the surface, the V's were pretty close together. At the depth of 250 feet the ore ceased; the arms of the V's came to a point. (A new ore-shoot has since been found at the 250-foot level, north.) The pockets of ore varied from 2 to 5 feet wide. The shaft was sunk on the vein to a depth of 590 feet, when the top of another pocket, or this time, perhaps, a genuine ore-shoot, was found, which, up to date, has proved to be 80 feet in length and about 60 feet deep, and is still good at the bottom (650 feet). The vein from 250 to 590 feet was well defined, varying in size from 4 to 6 feet, but carrying little value. This deep discovery should be regarded as a very encouraging feature in the camp.

I have a small specimen of ore reported to have been taken from a depth of 630 feet, which shows complete oxidation. The quartz which replaced the country-rock is clear and glassy.

Covering the quartz in one place is a deposit of kaolin, in which there are three small quartz crystals. Two of these contain in the center perfect hexagonal crystals of gold, minus the pyramidal termination; the third quartz crystal, doubtless when broken in the mine, or in transit, has lost its gold crystal. In the same specimen is another hollow shell of quartz, which was being gradually replaced by gold.

The Pharmacist vein, on the Zenobia claim, is faulted and thrown by a vein known as the Zenobia, having a N. and S. course. The throw was 21 feet at the depth of 14 feet, 12 and

Fig. 10.



Longitudinal Section of Vindicator Mine.

8 feet at the depths of 78 and 35 feet respectively. The Pharmacist vein after it leaves the Zenobia vein on its S. W. course ceases to be productive, and it was my impression early in 1893 that the cross-vein enriched the Pharmacist vein; but in view of the peculiar shape of the pockets and the later developments in both mines, such a theory has been discarded. The influence of cross-fissures in this camp has not been very marked either in the increased or decreased value of the mineral, in the change of dip or course of the main vein, or in the faulting of the vein (save in the above case); but the cross-veins, and they are legion, cross the older fissures with the same ease apparently as pencil-marks cross each other. The

rocks are not metamorphosed an iota. In the C. O. D. the influence of cross-fissures was mechanical, not chemical.

Vindicator.—The course of this vein is N. 47° W., the dip S.W. The ore-shoot as found in the 1st and 2d levels is 130 feet in length, and the pay-streak varies in width from 2 to 7 feet (the richer portions being found on the foot-wall) and pitches N.W. In the stope above the 2d level two kidneys of waste rock were found, as shown in Fig. 10.

At the 3d level, 196 feet deep, a second vein, striking N. and S., was discovered by cross-cutting to the east 22 feet from the shaft, and a shoot of ore was found in this vein, having a north pitch; the pay-streak being 4 feet wide and well oxidized.

Lillie.—This adjoining claim has an ore-shoot which dips N. on a N. and S. vein; the length of the ore-shoot at the surface being 100 feet, one shaft having been sunk on the south end of it, and another shaft on the north end, by lessees. The ore is only partially oxidized, although within 100 feet of the Vindicator shaft.

Victor.—This vein has a course of N. 45° W. in the main, yet for short distances it winds and deviates from a straight line. The dip is as irregular as the course, about 65° to the southwest. The size is no more amenable to rule than the course or dip, but varies from 6 inches to 8 feet. At a point 30 feet north of the old incline-shaft, at the 2d level, 86 feet deep, the vein forked, forming what are known as the "East vein" and the "West vein." Near the junction, on the 2d, 3d, and 4th levels of the East vein, were found the big deposits of ore (in one place 26 feet wide). The "West vein" is a larger vein, from 18 inches to 8 feet wide, but it does not carry such values as the "East vein."

At the surface there was but one ore-shoot; but at the 2d level the ore-shoot became divided into two, the north shoot dipping northwest, and the south shoot dipping southeast. The old incline-shaft had ore on both sides, but none in the shaft. The ore-shoot has been opened up for 1000 feet in length on the 4th and 5th levels.

The large bodies of ore are found at the south end, where the undeveloped part of the claim is. The vein, no matter what kind of rock encloses it, whether phonolite, andesite, or

even tuff, maintains its value, save at the south end, where, as it reaches the surface in the breccia, it is unproductive. The same remark applies to the Independence and Portland mines in this camp, and partially answers Prof. Wm. P. Blake's questions in his paper on "Gold in Granite and Plutonic Rocks" presented at this meeting (see *ante*, p. 293). The change of country-rock has had but little influence on the veins here as yet. But this is a young camp, only four years old, and later developments may prove the contrary.

Buena Vista.—This is a continuation of the Victor vein, and exhibits approximately the same conditions.

Lucky Guss.—The strike of the vein is N.W., the dip to the east about 81°, the width of vein 3½ feet. The pay-streak varies from 2 to 12 inches, and is always found on the foot-wall. The ore-shoot, at present worked at the fifth level, is exposed for 18 feet in length, and is still being developed. It pitches 45° to the north. The property has just come into the hands of a new company, after being worked by lessees. The lessees had three shoots, varying in length from 8 to 30 feet, all having a northerly pitch.

A new vein was recently discovered at the surface in this claim, right in the phonolite. The walls are phonolite, the vein is phonolite, and the only substance around not phonolite is the mud which carries the gold, two tons of which, extracted in five days' sinking of the shaft, sampled 92 ounces gold per ton. The shaft is now 40 feet deep, and a drift is about to be extended north. This appears to be a parallel vein.

Battle Mountain.

Portland.—The Portland main ore-shoot was caused by a junction of three veins, one having a N.W. course, the second a N.E. course, the third a small cross-vein known as the Sylvanite vein. The size of the ore-shoot varies from 12 to 30 feet in width, and it has pinched to 6 feet; its length varies from 175 to 225 feet. Until a depth of 400 feet was reached, the shoot pitched S.W. at an angle of 52° near the surface. At 400 feet, in the granite, the ore-shoot straightened, and at 600 feet there is a dip to the north of 85°, but without any perceptible effect on the value of the ore-shoot, or on the coun-

try-rock. There are values in the vein from Black Diamond to Anna Lee, a distance of about 800 feet.

Independence.—This vein has about a N. and S. course, sometimes varying from N. 3° E. to N. 20° W., and dips to the east from 85° to 90°. The pitch of the ore-shoot is to the N. 35°. It is about 12 feet wide and 350 feet in length, the remainder of the vein being of low grade.

Beacon Hill.

Prince Albert.—The upper portion of Beacon hill consists of a large mass of phonolite, overlapping the granite. The course of the Prince Albert vein is N.E. and S.W., and it dips from 80° to 90° S.E., or towards the gulch. The dike or vein is 80 feet wide, and has three ore-shoots, having the same dip and course as the vein. These ore-shoots, two of which have a length of 50 feet, and one of 30 feet, are 3 feet wide, and yield ore of the value of about $70 per ton. Some $200,000 has been extracted above the 120-foot level. The rich ore is always found on the hanging-wall.

At the depth of 90 feet, the vein or dike (the words are used advisedly, it being difficult, according to Prof. Kemp's definition, to say to what class some of the ore-deposits here as well as elsewhere are to be ascribed) encountered another dike 12 feet wide, apparently intrusive, having a dip into the hill of 25° N.W. The vein continued through this dike only as a small rusty seam, but followed on the top of the dike for 12 feet, actually shown in the workings, and presumably about 30 feet further, when it touched the granite and straightened itself. The value of the vein 5 feet wide on the top of the intrusive dike was from $5 to $8, but on the granite the value increased to $12 and $15 per ton.

At a depth of 130 feet, the ore-shoots are getting longer, and it also looks as if the ore-shoots were pitching N.W. instead of S.E. The ore in the granite is difficult to distinguish from the granite itself, save by the substitution often of iron pyrites for the mica. Cross fissures have exercised no influence on the vein. The line of superficial alteration is at the surface; there are but small quantities of oxidized or "red" ores.

*

The Phosphate-Deposits of Arkansas.

BY JOHN C. BRANNER, STANFORD UNIVERSITY, CAL.

(Colorado Meeting, September, 1896.)

Stratigraphic Position of the Deposits.—During the progress of the geological survey of Arkansas, in the northern part of that State, it was found that the interval between recognizable Lower Silurian rocks below and recognizable Lower Carboniferous rocks above contains no representatives, or but poor representatives, of the whole Upper Silurian and Devonian series. In some places one passes in the space of a foot from Lower Silurian rocks to the Lower Carboniferous—the Boone chert of the Arkansas survey, or the "Burlington" beds; at other places this interval is 3 or 4 feet thick, but it is generally concealed by the ready decay of the black shale—the Eureka shale—which fills it, so that for a long time it was passed over undetected. It was only after considerable work in the region lying north of the Boston mountains, and by the valuable palæontologic work of Dr. Henry S. Williams, of Yale University, who kindly accompanied me on a trip through this part of the State, that the importance of this interval came to be recognized, and the thin beds occupying it to be critically studied. It then appeared that this interval was occupied, for the most part, either by a greenish or black shale or by a sandstone; and eventually the sandstone was found to have a maximum thickness of 40 feet on South Sylamore creek in 15 N. 11 W., Section 21. It was, therefore, called Sylamore sandstone.

The shale of this interval is generally from 3 to 10 feet thick; it is a constant feature of the geology through Carroll county, and is well exposed in the streets and cuts in and about Eureka Springs, and was for this reason named the Eureka shale. Its maximum thickness is 50 feet in the northwestern part of Benton county.

The relations to each other of the Sylamore sandstone and the Eureka shale I have never determined satisfactorily; for

generally when one is present the other is absent, while on South Sylamore creek, where both seem to occur, the outcrops are so concealed by the decay of overlying beds that their relations cannot be seen.

Analyses of the phosphatic nodules from the Sylamore sandstone are given below. The stratigraphic relations of the phosphates to the Eureka shale were so intimate that I had analyses made of both the black and green varieties. There is, however, no phosphate-bed at the point where these samples were collected.

Analyses of Eureka Shale from Dairy Spring, Eureka Springs, Ark.

	Black Variety. per cent.	Green Variety. per cent.
Silica, SiO_2,	59.33	64.28
Alumina, Al_2O_3,	15.51	11.87
Iron oxide, Fe_2O_3,	7.57	9.60
Pyrites, FeS_2,	3.48	4.32
Lime, CaO,	1.05	0.99
Magnesia, MgO,	0.79	0.55
Potash, K_2O,	5.22	3.12
Soda, Na_2O,	trace.	trace.
Loss on ignition,	7.19	4.42
	100.14	99.15
Water at 110° to 115° C.,	1.31	2.76

The Marble report of the Arkansas Survey, Plate X., opposite page 212, gives a group of sections across the northern part of the State, showing the varying thicknesses of the Sylamore sandstone and the Eureka shale.

Mr. Hopkins has the following on the Sylamore sandstone :*

"The Sylamore sandstone is generally an insignificant bed, being often but a few inches in thickness and readily disintegrating, so that it is frequently overlooked even when present. It differs greatly in character in different parts of the area; in some places it is made up of rounded grains of hard crystalline quartz, interspersed with black, rounded, irregular pebbles; in other places it is a soft earthy rock of a yellowish brown color; in still other places an arenaceous shale. The dark-colored pebbles are a peculiar feature of this rock, yet they are not always present."

The Eureka shale is clearly the Arkansas equivalent of the Tennessee bed called by Safford the "Black shale." In Tennessee this shale was considered as Devonian, chiefly on account

* *Ann. Rep. Geol. Sur. Ark.*, for 1890, vol. iv., "Marbles and Other Limestones," by T. C. Hopkins, Little Rock, 1893, pp. 212–213.

of its position between the Silurian and Lower Carboniferous; for no characteristic Devonian fossils have thus far been found in these shales, either in Tennessee or in Arkansas. In the latter State, *Lingula*, *Discina* and crinoid stems are the principal fossils found, and these have no diagnostic value. Other brachiopods occur in this shale near Batesville, but they have not yet been reported upon by an expert palæontologist.

The intergradation at some points of the shale with the overlying limestone, and the occasional presence in the upper part of the shales of the crinoid stems that form the mass of the limestone, have led me to think the shale might belong with the overlying Lower Carboniferous beds. So far as the objects of the present paper are concerned, however, the precise age of these beds is a matter of no great importance, except in so far as it enables one to locate them. The position of these strata is constant when they are present at all, and they are readily found by any one acquainted with the rocks and stratigraphy of northern Arkansas.

The Sylamore sandstone and the Eureka shale are mentioned here simply to point the way to the phosphate-beds of the State; for it is in, or associated with, these rocks that the phosphates occur.

This will be plainer after the known deposits are described. It will be seen also that these phosphates occupy the same stratigraphic position as do the Tennessee beds. Through the kindness of Mr. Hayes I have been enabled to examine specimens of phosphate-rocks from Tennessee, and Mr. Hayes's description of those deposits* shows that there are deposits in that State at horizons other than the Devonian interval.

In the State of Arkansas I have never seen Palæozoic phosphate-deposits known to be elsewhere than in this interval. The precise geologic position of the beds north of Hot Springs is not known. They are Lower Silurian, however.

I learn from Dr. Safford, of Nashville, of a late discovery of an important bed of phosphate rock in what he calls the Capitol limestone, which is the equivalent of the St. Clair marble of Arkansas.† Several analyses have been made of the St. Clair

* "The Tennessee Phosphates," by C. W. Hayes, 16*th Ann. Rep. U. S. Geol. Sur.*, Part IV., pp. 610–630.

† These Tennessee beds are described in the *American Geologist* for Oct., 1896, pp. 261–264.

marble, showing that rock to contain considerable phosphoric acid, but never enough to suggest its having any economic value.

The general geologic section north of the Boston mountains is as follows:

Coal Measures of Pennsylvania in the top of Boston Mountains and their Outliers.

		Maximum thickness. Feet.
Lower Carboniferous or Mississippian.	Boston group (shales, limestones, sandstones, cherts),	780
	Batesville sandstone,	200
	Fayetteville shale,	300
	Boone chert and limestone,	370
Devonian (?) (Phosphate horizon.)	Sylamore sandstone,	40
	Eureka shale,	50
Lower Silurian.	St. Clair marble,*	155
	Izard limestone,	285
	Saccharoidal sandstone, Magnesian limestones, Sandstones, cherts, etc.,	1750+

By means of this section any one at all acquainted with the geology of the region can readily find the interval at which the phosphate-beds are to be looked for.

The accompanying map will also aid any one to locate these deposits approximately, and a little search on the ground will soon discover them.

The phosphate-deposits occur in a hilly region. The horizontal rocks have been trenched by gorges from 50 to 500 feet deep, along whose sides the various rocks outcrop. These hills, valleys and slopes are heavily timbered, except in certain parts known as the "flint hills," where the soil is thin and rocky, and the timber rather stunted in growth.

Appearance.—The phosphate pebbles vary greatly in size, color, and form. Sometimes they are no larger than a pinhead, and from this they vary up to the size of a hen's egg, and are even larger. They are sometimes gray, often of a light yellowish-brown or buff color (and where the rock is made up almost entirely of the nodules they are generally of this buff color), and they are also often quite black. The large black ones have often been found to contain fossils, especially *Lingula.*

* Dr. Henry S. Williams says the upper part of the St. Clair is Upper Silurian. *Am. Jour. Sci.*, 3d ser., 1894, xlviii., 326.

These black nodules are hard and rock-like, and break like a compact limestone. The lighter-colored ones are more earthy, possibly on account of weathering. In form they are very irregular, are frequently pitted or indented over the outer surface, and look like so many small Irish potatoes; often they are flattened or oblong. Mr. Hopkins makes the general remark on these nodules that they are less abundant in the thicker beds of Sylamore sandstones, and more abundant in the thin ones. This, however, is not always the case; some of the thin beds are quite as poor in phosphates as the thick ones, while some of the richest beds are fifteen feet in thickness. The sand-grains found in the sandstone in which these nodules occur are sometimes so smooth and rounded that they look like so many little pearls.

Origin of the Deposits.—The origin of the phosphate-beds of Arkansas cannot be comprehensively discussed here, and it is not likely that such discussion, even if it were possible, would add much to our knowledge of the distribution of the beds.

In any search for such deposits in this region, the following leading features are of importance:

1. That they occupy the geological horizon of the Devonian rocks.

2. That they are, in the main, conformable with the rocks both above and below them.

3. That they vary much in thickness and character.

The fact that Upper Silurian and Devonian time is here represented by a very thin deposit compels us to believe, either that the region was not under water in Devonian time, or else that, being under water, it was so far from the shore and the water was so deep, that almost no sediments were laid down over this part of the sea-bottom during this long period. I am not disposed to believe that these deposits were made as surface-accumulations; the regular and undisturbed condition of the upper surface of the underlying beds across a region a hundred miles wide in northern Arkansas, is as nearly perfect as we are accustomed to find it in any sedimentary deposit. A land-surface so smooth and regular is out of the question. We are, therefore, reduced to the necessity of believing that this interval, with its phosphate-deposits, represents the slow accumulation of organic matter over a comparatively deep sea (not

abysmal, however) during the Upper Silurian and Devonian periods.*

Local Details.—The most easterly exposures of the phosphate-beds now known are in the vicinity of Hickory Valley, in Independence county. A systematic search for the line of contact between the St. Clair marble and the Boone chert beds, may lead to the discovery of these beds somewhat further east. They will not be found east of Black river, however, for the Palæozoic beds are cut off just west of that stream, and are overlapped by the Tertiary and later beds that cover all eastern Arkansas.

The deposits near Hickory Valley are at Mr. Milligan's place, twelve miles northeast of Batesville, in 14 N., 5 W., Sec. 6. The rock looks like a conglomerate of pudding-stone, and is exposed at the mouth of a cave in a hillside, where it forms a bed 15 feet or more in thickness. The bed rests upon the St. Clair marble, but the top of it is concealed by the chert *débris*.

This rock weighs about 150 pounds to the cubic foot as it lies in the ground; and, if it is of even grade, it should yield a ton of rock to the square foot of the surface over which it has a thickness of 15 feet.

Analyses of Phosphate-Rock from the Milligan Place, near Hickory Valley, Independence County.

	Per cent.	Per cent.
[Samples from inside of the cave.]		
Phosphoric acid, P_2O_5,	26.13	26.77
Equivalent to bone phosphate, $Ca_3P_2O_8$,	51.53	52.08
Iron and alumina,	5.89	5.86
[Second set of samples from inside of the cave.]		
Phosphoric acid, P_2O_5,	29.40	29.98
Equivalent to bone phosphate, $Ca_3P_2O_8$,	64.82	65.37
Iron and alumina,	3.08	3.87
[Samples from boulders outside of cave.]		
Phosphoric acid, P_2O_5,	31.06	31.11
Equivalent to bone phosphate, $Ca_3P_2O_8$,	67.21	67.74
Iron and alumina,	8.01	7.05
[Samples from fragments outside of cave.]		
Phosphoric acid, P_2O_5,	33.54	33.86
Equivalent to bone phosphate, $Ca_3P_2O_8$,	72.89	73.28
Iron and alumina,	5.19	5.09

* For discussion of the conditions of accumulation, see C. W. Hayes, in *16th Ann. Rep. U. S. Geol. Sur.*, Part IV., pp. 620–623.

Mr. T. C. Hopkins, in his work upon the marble in the region about Batesville, Independence county, makes mention of a bed 2 inches thick, "of impure siliceous material, containing black pebbles and numerous white calcite fossil brachiopods," in the bluff 100 yards below the fourth ford on the Polk Bayou road from Batesville. This bed was looked for by me, but never found, probably on account of its thinness, and also because, both up stream and down stream, the parting is concealed by *débris*. The sample sent in is of a greenish color and the phosphate nodules are black.

About Cushman the phosphate-rock is usually of a light-brown color, but is darker on a weathered surface than in a freshly-broken one. This phosphate-rock is common in the manganese mines, and there is no doubt that the low grade of some of the Batesville manganese-ores is due to the presence of these phosphate-beds in the manganese-region. In the Arkansas Survey's report on manganese, Dr. Penrose says:*

"At the southern mine, near Cushman, there are frequently found, at the base of the chert and just above the residual clay which has resulted from the destruction of the St. Clair limestone, a red, brown, or mottled seam of a dry, earthy consistency. It is known by the miners as "ocher," and may possibly represent the former contact of the chert and St. Clair limestone. Elsewhere in the neighborhood of Cushman, and thence westward to Lafferty creek, there are frequently found loose fragments of a soft, buff-colored or rusty-brown material of an earthy or fine sandy structure, frequently containing thin ferruginous seams or flat concretions."

This is the phosphate-bed, though its true nature was not suspected at the time the manganese report was written. Indeed, this bed at other points had suggested to the writer that the material was volcanic tuff; and Dr. J. E. Wolff, of Harvard, after a microscopic examination of it, had rather encouraged this view.†

About three-quarters of a mile south of Cushman, on the wagon-road to Batesville, phosphate-rock with some manganese is exposed in the road. It is here between 10 and 20 feet thick, judging from the fragments to be seen. The phosphate-rock is probably exposed at this elevation on account of a low anticlinal fold.

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* *Ann. Rep. Geol. Sur., Arkansas*, 1890, vol. i., "Manganese," by R. A. F. Penrose, Jr., p. 126.

† "Manganese," by R. A. F. Penrose, Jr., p. 127.

In 14 N., 7 W., Sec. 8, northwest quarter, fragments of the phosphate-rocks cover the hillside nearly opposite Farrell's cave. The details of the geologic action are concealed; but, judging from the fragments in sight, the phosphate-rock is 20 feet or more in thickness, and rests upon the St. Clair marble bed. These fragments are angular, and are scattered along a contour for about 250 feet; none of them were seen in place.

About half a mile west of Cushman, on what is known as the Meeker place (14 N., 7 W., Sec. 8), the phosphate-bed is exposed in an old manganese-pit on the north slope of the hill, and from this point may be traced westward, along the north face of the hills, for 2½ or 3 miles. It always overlies the marble, and, as the rocks are horizontal, the position of the bed is easily determined, even when it is concealed, as it is for the most part. Along the contour followed by the phosphate-beds, a good many pits have been dug in search of manganese, and in these pits the phosphate-beds are almost always uncovered. These exposures are interesting also as showing the relation of the manganese-beds to the other rocks—a subject that has led to some discussion.*

The section in one of the pits on the Meeker place is as follows, beginning above:

Soil at the top.	Feet.
Black phosphate rock,	2.5
	Inches.
Manganese-ore,	1 to 3
	Feet.
Marble,	3

This exposure is about 50 feet in length.

Another one of these pits has

Soil at the top.	Feet.
Chert,	3
Phosphate-rock,	3.5
Iron and manganese,	2
Concealed,	2
Marble,	3

A third pit has

Soil at the top.	Inches.
Phosphate-rock exposed,	18
Iron and manganese,	8
Marble,	?

* *Am. Jour. Sci.*, 3d Ser., xlviii., 325–331.

A fourth pit has

	Feet.
Chert fragments and soil at the top.	
Phosphate-rock,	3.5
Manganese, iron and phosphate-rock,	3
Underlying rock not seen.	

Analysis was made of an average sample from these pits, the mass of the rock being used and not the nodules alone. The following was the result:

	Per cent.
Phosphoric acid, P_2O_5,	22.62
Equivalent to bone phosphate, $Ca_3P_2O_8$,	49.38
Iron and alumina,	8.82

Dr. Penrose reports* for 15 N., 8 W., Sec. 29, northwest quarter:

"A rock which, though it was only seen in loose boulders, probably belongs at the contact of the chert and the St. Clair limestone. It is a dark brown or black, earthy, siliceous, soft rock, probably about 1 foot in thickness and emitting a strong odor on a fresh fracture."

This is the phosphate-rock again.

Rock of the same general character as the Cushman phosphate-beds is said to occur at several other places west of Cushman, notably at the following: 14 N., 8 W., Sec. 12, middle of the section, along the north slope of the valley; Sec. 11, south half, about the head of the hollow draining into West Lafferty creek (Mr. Hopkins reports small black pebbles in this rock); and section 10, on the west side of the valley slope toward West Lafferty creek, and at the southwest corner on the side of the hill facing White river; Section 9, at Penter's Bluff in the northeast corner of the section on the side of the hill sloping toward White river. The boulders of the phosphate-rock (Sylamore sandstone), found in the ravines and along the creek-beds in this section of the country, lead to the conclusion that the bed is often 3 or 4 feet in thickness. The rock, however, is harder than that in the Lafferty creek valley, where it is soft and yellow.

Mr. Hopkins noted the following sections 300 yards northeast from John Greenway's house, in 14 N., 9 W., Sec. 3:

* Manganese report, p. 126.

	Feet.
St. Joe marble,	8
Light-green siliceous rock,	2
Concealed,	3
St. Clair marble,	10

I take the green rock of this section to be the phosphate-bed, which, at this place, on account of the underlying rock being concealed, may have a greater thickness than is reported above.

Another exposure in the same section, on the east side of Cagin creek, opposite Jeff. Greenway's house, about half a mile up the creek from John Greenway's, is as follows (Hopkins):

	Feet.
Chert,	20
Limestone,	1
Green shale,	$\frac{1}{4}$
St. Joe marble, the bottom layers containing patches of black shale, sand and black pebbles,	8
Green and black shale (bottom concealed),	15

The black pebbles below the St. Joe marble here are the phosphate-layer. At the upper end of this bluff, this layer is, in places, 8 inches thick.

In 15 N., 9 W., Sec. 32, on the north side of Dry creek, Mr. Hopkins found "calcareous and siliceous nodules in a siliceous limestone overlying the St. Clair marble;" and again, in 15 N., 10 W., Sec. 22, west side of Little Rocky bayou, he found at the same horizon a bed of sandstone "containing numerous black ferruginous nodules," no doubt the phosphate-nodules. He does not mention its thickness.

The next region in which the phosphate-bed has been observed is along Sylamore creek and its affluents.

On Roasting Ear creek Mr. Hopkins found at Norman's, in 15 N., 13 W., Sec. 1, what, I have no doubt, is the phosphate-bed. His notes state that it is between the "St. Joe marble above and the St. Clair marble," and that it contains black nodules. He does not mention the thickness.

On the west side of Sylamore creek, opposite the mouth of South Sylamore, the limestone below is exposed to within 3 feet of the Boone chert, and the space between is concealed by *débris*. In the stream-bed below, however, are fragments of Sylamore sandstone containing phosphate-nodules. These fragments must have come from the horizon here concealed.

At the little school-house near the mouth of Lick fork, the contact between the Silurian and the Lower Carboniferous rocks is exposed in the rear of the building on the side of the hill near the creek. The bed between them is about 18 inches thick, the upper part of which is sandy and contains black phosphatic nodules and the lower half decays to a soft earth, resembling the Cushman phosphate-beds.

On the south side of Sylamore creek, about half a mile below the mouth of Lick fork, there is a similar concretionary bed 1 foot or more in thickness, between the St. Clair and the St. Joe marbles (Hopkins). The sample collected shows this to be the dirty brown phosphate-rock.

On the west side of Sylamore creek and south of Dry creek, the Sylamore sandstone seems to have a thickness of about 15 feet, 6 feet of which is well exposed at one place high on the side of the hill. The rock at this place is chiefly a rather coarse-grained, hard sandstone, almost a quartzite. There are some layers 2 or 3 inches thick that are rich in phosphate-nodules, but most of the bed is poor.

In 15 N., 11 W., Sec. 21, in the hill southeast of Prater's, the base of the Sylamore sandstone is about 100 feet above the stream and it seems to have a thickness of 30 or 40 feet. It is hard and compact where examined, however, and the phosphate-nodules are not abundant.

In 16 N., 13 W., Sec. 24, the Sylamore sandstone is exposed near the top of the hill south of Capp's fork. Mr. Hopkins says fossils are found in the sandstone at this place. Above it is the Boone chert, and below it the Izard limestone.

On Long creek, Searcy county, the interval between the Silurian and Lower Carboniferous rocks is occupied by a coarse-grained sandstone, containing phosphate-nodules. The bed seen at this place has a thickness of only 2 feet; it may have a greater thickness in this region.

In the region known as "The Narrows," in 17 N., 15 W., Sec. 13 (or 24), on the west side of the road leading from Rush P. O. to Big creek, there is a nodular sandstone, lying between the St. Joe marble above and the Izard limestone below. This bed is only 2 or 3 inches thick at this place. About 200 yards south of this last section, about 2 feet of the bottom layers of the red St. Joe marble contain black pebbles (Hopkins), which are in all probability phosphate-nodules.

On Dry creek in 16 N., 16 W., Sec. 30, on the west side of the creek, is the following section (Hopkins):

Greenish-yellow shale with black pebbles at base.

Lead-colored shale with black pebbles.

Arenaceous, pyritiferous shale.

St. Clair marble.

At another outcrop near this section the sandstone rests directly on the St. Clair marble.

Mr. Hopkins reports also an exposure of Sylamore sandstone 20 feet thick in 16 N., 16 W., Sec. 18, on the south side of Tomahawk creek, west of the Marshall-Yelville road; he makes no mention of its containing pebbles.

Near the middle of this same Sec. 18, in the field above Baker's gin, fragments of phosphate-rock are scattered through the soil overlying the St. Clair marble. This material is said to have been dug through, and to have a thickness of 4 or 5 feet.

An analysis of a specimen of this phosphate-rock shows it to have the following composition:

	Per cent.
Phosphoric acid, P_2O_5,	35.11
Equivalent to bone-phosphate, $Ca_3P_2O_8$,	76.62
Oxides of iron and alumina,	7.21

On Rush creek (17 N., 15 W.) there is a fault along which the stream flows. North of this fault the contact between the Carboniferous and Silurian rocks is in the hilltop; and if there is any phosphate-bed in this contact-zone, it is so thin that it was overlooked in the several sections I made across it. On the south side of the valley—the down-throw side of the fault—a bed of sandstone, probably the Sylamore sandstone, is enclosed between two marble beds.

At St. Joe, the contact between the Carboniferous and Silurian rocks is exposed about three-quarters of a mile above the village, in the bluff where South Mill creek enters Monkey run.

The section at this place is:

	Feet.
Chert and cherty limestone (Boone chert),	35
Light-colored marble,	10
Pink marble (St. Joe bed),	40
	Inches.
Sandy conglomerate,	4

	Feet.
Olive-green shale with phosphate-nodules,	2
Sandy conglomerate with nodules,	3
Concealed,	4
St. Clair pinkish marble,	5

This last bed is 25 feet in all, and below it is the blue Izard limestone. Analysis of the black nodules from this section showed them to contain:

	Per cent.
Phosphoric acid, P_2O_5,	31.75
Equivalent to bone-phosphate, $Ca_3P_2O_8$,	69.31
Oxides of iron and alumina,	8.10

Just north of this exposure, the St. Joe fault carries this line of contact to near the hilltop, over 200 feet higher. In this hilltop the contact-beds were not observed.

On the southwest side of Monkey run, where the St. Joe and Harrison road turns to the right and ascends the hill, the parting is indicated by a bluish olive-green bed, about 3 feet thick, containing phosphate-nodules. Analysis of the brown phosphate-nodules from this bed shows:

	Per cent.
Phosphoric acid, P_2O_5,	25.92
Equivalent to bone-phosphate, $Ca_3P_2O_8$,	56.58
Oxides of iron and alumina,	9.01

This bed is mentioned by Owen in his first report; but he did not suspect its true nature, for he speaks of it as a "vein of ore containing iron and manganese."*

The Sylamore sandstone on Cave creek is exposed at the mouth of "Saltpeter Cave," in 15 N., 19 W., Sec. 27. I have not myself visited this locality; and the following description is taken from notes made by Mr. Hopkins while working up the marble beds.

The space between the St. Joe above and the St. Clair marble below is occupied by a bed of shale, 12 to 14 inches thick. At the bottom this shale is siliceous, black and lumpy; and similar nodules are scattered through the St. Joe marble for a foot or more from the bottom.

On the north side of Little Buffalo, in 16 N., 21 W., Sec. 26, 200 yards above the third ford below Jasper, is an exposure

* *First Report of a Geological Reconnoissance of the Northern Counties of Arkansas*, by D. D. Owen, Little Rock, 1858, p. 79.

of shale that contains pebbles resembling Sylamore sandstone (Hopkins).

In 16 N., 21 W., Sec. 9, on the north side of Big Buffalo and 200 yards above the mouth of Sneed's creek, the Sylamore sandstone is from 2 to 6 inches thick and contains black pebbles (Hopkins).

At Elixir Springs, Boone county, 20 N., 19 W., Sec. 36, the Sylamore sandstone is 2 inches thick. It lies between the St. Joe marble above and a light gray limestone below (Hopkins).

In 16 N., 26 W., a quarter of a mile south of the southeast corner of Section 1, Prof. J. F. Newsom found a bed of limestone conglomerates, the materials of which strongly resemble phosphate nodules. It was analyzed and found to contain only about 8 per cent. of calcium phosphate.

My notes upon the zinc-deposits contain many references to the phosphate-beds at the contact between the Boone chert or St. Joe marble and the Lower Silurian rocks. This bed is exposed in many of the shafts, drifts and prospect-pits through Marion and Boone counties; for the zinc is frequently found just below the St. Joe marble.

The Ouachita Phosphate-Region.—The study of the Silurian rocks of the State never having been completed by the Geological Survey, it has not been possible to correlate the stages north of the Boston mountains with those of the Silurian area of the Ouachita uplift or great anticline belt that crosses the State from south of Little Rock to Dallas, Polk county.

In this region the Lower Coal Measures or Pennsylvanian rocks rest upon Silurian rocks, the Lower Carboniferous or Mississippian beds, so characteristic of the northern part of the State, being altogether wanting as far as known. Conglomerate beds are known, however, in the Ouachita uplift, which bear so striking a resemblance to the Sylamore sandstone that they can scarcely be distinguished in hand-specimens. The only apparent difference is that the Ouachita-uplift specimens appear to be a little more crystalline or metamorphosed. On account of the crushing of the rocks throughout this southern area this was to have been expected.

Mr. Griswold, who reported on the novaculites of this region, sent in a specimen of this phosphatic sandy conglomerate from

1 S., 19 W., Sec. 6, southeast quarter, on the Hot Springs and Fort Smith road. These rocks have a northward dip of 75° and a thickness of about 200 feet. A similar conglomerate is reported from 1 N., 20 W., Sec. 35, northwest quarter.*

As the rocks throughout the region of the Ouachita uplift are much folded, the distribution of a phosphate-bed will not be so easily traced as in the region of horizontal beds in the Boston mountains region.

In the region of Lower Coal Measure rocks in the vicinity of Amity, Clark county, a conglomerate found by Dr. George H. Ashley resembles the phosphate-rock somewhat; but a chemical examination of one example of it showed it to contain only about 9 per cent. of calcium phosphate.

Cretaceous Phosphates.—Bands of black phosphate-like pebbles are known in some of the Cretaceous beds of the State. One of these first noticed by the writer in 1887 was afterwards referred to as a "deposit of blackened quartz pebbles."† I find, on making a chemical examination of these pebbles, that some of them are chert and some of them are phosphate nodules.‡ I do not know of their occurring in the Cretaceous rocks in sufficient quantities to make them valuable.

Attention should be directed in this place to another possible source of phosphates in Arkansas—the Cretaceous marls of the southwestern part of the State. These marls are very like the greensand marls of New Jersey, and are, in all probability, available for the same purposes as those for which they have long been so extensively and successfully used in New Jersey, namely, for direct application as fertilizers to the soil. No thorough practical test, however, has ever been made of the Arkansas greensands.

In France similar greensands are now used for the manufacture of high-grade fertilizers, and it is suggested that possibly the Arkansas beds may be handled in the same way. At Bellegarde-sur-Valserine, Ain, France, the greensand-beds are

* Postscript.—A chemical examination of this rock has shown that it contains so little phosphoric acid that it is doubtful whether it should have been mentioned. This conglomerate, however, ought to be further examined for phosphate-rock.

† *Ann. Rep. Geol. Sur. of Arkansas* for 1883, vol. ii., p. 77.

‡ For specimens of the Cretaceous pebbles I am under obligations to Mr. Rupert B. Haskins, Instructor in Chemistry in the Arkadelphia Methodist College, Arkadelphia, Ark.

exposed in horizontal beds from 4 to 6 feet thick on the banks of the river Rhone. These beds are mined by drifts; the crude material is run through a crusher, and thence passes through water into ordinary revolving wire-screens, which allow the fine sands and worthless matter to be washed out, while the phosphate nodules which are hard enough to resist crushing in the rollers fall out at the ends and are afterwards used as rock-phosphates for the manufacture of high-grade fertilizers.

I regret that I cannot at present give the areal distribution of these greensand marl-beds in Arkansas. They are, however, confined to the Cretaceous rocks; and the general distribution of the rocks of that age is shown on the geological maps accompanying the report of the Geological Survey of Arkansas on the Mesozoic and Tertiary.*

I have noted the greensand-beds at Washington, Hempstead county, where they are exposed between the railway station and the Garland place; also on the Haskins place on the Little Deciper, about three miles southwest from Arkadelphia.

Value of the Phosphates.—The data at hand regarding these Arkansas phosphate-beds are not sufficient to determine whether or not they exist at many points in quantity and of a quality that will make them of commercial value.

Most of the facts in my possession were gathered before the nature of the deposits was known, and when they were not considered to have any economic value. After ascertaining their nature, I offered the State of Arkansas my services in working up the subject in detail, but the proposition was not acted upon. To make a careful examination of the entire outcrop across the whole Palæozoic area of North Arkansas, and through the Silurian area of the central part of the State, would cost more than private parties would, as a rule, be willing to venture in such an enterprise. These facts should be borne in mind in any attempt, from the data here given, to arrive at decided conclusions regarding the commercial value of the deposits. No doubt, after the geology and the appearance of the phosphate-rocks become known throughout the region in question, the existence of the more important beds will gradually come to light, and means can be devised for their exploitation. Several railway companies have diligently sought valid reasons for

* Vol. ii. for 1888 and vol ii. for 1892.

extending their lines into this North Arkansas region; it is hoped that the knowledge of these phosphate-beds may offer some of them the inducements they have so long looked for.

The few analyses made of the phosphate-rocks are here brought together for purposes of comparison, and to indicate the market-values of the materials.

Analyses of Arkansas Phosphate-Rocks.

LOCALITY.	Phosphoric Acid, P_2O_5.	Equivalent to Bone Phosphate, $Ca_3P_2O_8$.	Oxides of Iron and Alumina.
	Per cent.	Per cent.	Per cent.
Milligan place, specimens from inside of cave	26.13 26.77	51.53 52.08	5.89 5.86
Milligan place, second set of inside specimens	29.40 29.98	64.82 65.37	3.08 3.87
Milligan place, specimens from boulders on surface	31.06 31.11	67.21 67.74	8.01 7.05
Milligan place, specimens from small surface fragments	33.54 33.86	72.89 73.28	5.19 5.09
Meeker place, near Cushman	22.62	49.38	8.82
Monkey run, black nodules	31.75	69.31	8.10
Baker's gin, buff nodules	35.11	76.62	7.21
S. W. of Monkey run, brown nodules	25.92	56.58	9.01

Cost.—As no attempt has ever been made to work the Arkansas phosphate-deposits, the probable cost can only be estimated from the cost of mining similar deposits in Tennessee and from the price of labor in Arkansas. Mr. Memminger puts the cost of the Duck river material on board cars at $2.75 per long ton. This is distributed as follows:

	Per ton.
Mining,	$1.00
Hauling 9 miles to railway,	1.50
Breaking,	0.25
Total,	$2.75

The cost of a long ton at the Tennessee and Southern Company's mines he distributes thus:

	Per ton.
Mining,	$1.0
Hauling 4 miles to railway,	1.00
Breaking,	0.25
Total,	$2.25

This rock paid $2.91 freight to Atlanta, and was sold for $6.00 a ton.*

* "Commercial Development of the Tennessee Phosphate," by C. G. Memminger, *16th Ann. Rep. U. S. Geol. Sur.*, Part IV., pp. 631–635.

Teams can be had in the manganese-region of Arkansas for from $2.00 to $3.00 a day; and, by contract, the phosphate-rock from the deposits from 1 to 2 miles west of Cushman could be put on board cars at a cost of 50 cents a ton or less. Other deposits would cost more for teaming on account of greater distance from the railway. Freight from Cushman to St. Louis is about $2.25 a ton. The cost per long ton would therefore be about as follows:

	Per ton.
Mining,	$1.00
Hauling,	0.50
Breaking,	0.25
Freight to St. Louis,	2.25
Total,	$4.00

In Tennessee beds averaging 30 inches thick are worked, and the rock is hauled 5 miles to the railway with a profit.

The Duck River Company, 9 miles east of Centerville, Hickman county, Tenn., has stripped to a depth of 20 feet to uncover a phosphate bed 36 inches thick.*

Unskilled labor can be had at as low rates in Arkansas as in Tennessee, while living in the phosphate-region is cheap and good, and the region is high and healthful and well supplied with excellent water.

Method of Mining.—Inasmuch as the phosphate-beds of north Arkansas are almost invariably in horizontal beds between two beds of hard rock, and as they are for the most part exposed on the sides of gorges or valleys where erosion has cut down through these beds, whatever mining is done will be in drifts. The overlying rock, generally the St. Joe marble, is sufficiently massive and coherent to make a good roof, but if timbering should be necessary, the best of timber for such purposes covers the entire region, and can be had at little cost.

The only place I now know of at which the phosphate-rock can be had at the surface or by stripping, is over a limited area at Baker's gin in 16 N., 16 W., Section 8.

Possibly some method can be devised by which the black pebbles can be separated from the sandy portion of the Sylamore sandstone. The character of these nodules is shown by the analysis from Monkey run to be very good; the rock as a

* C. G. Memminger in 16*th Ann. Rep. U. S. Geol. Sur.*, Part IV., p. 631.

whole can probably not be used, except where the pebbles are unusually abundant.

Transportation.—One of the greatest difficulties at present in the way of any development of the phosphate-deposits of the State is the absence of prompt and cheap transportation-facilities. It is true that the White river flows through the phosphate-region; but that stream furnishes an uncertain and unsteady means of shipment. Moreover, freight would have to be transferred to the railway either at Batesville or at Newport, or it would have to make the long, tedious voyage down to the mouth of White river, and thence up the Mississippi to Memphis or St. Louis or down to New Orleans.

The deposits now known nearest to the railway are those in the vicinity of Cushman; and it is for these convenient deposits only that the favorable rates given above are available.

If the disadvantages of transfer from boat to railway and high railway-rates were removed, there would still be a considerable, and, at present, uncertain cost of hauling from the remote mines to the boats, always over bad country roads.

Market.—In case any of the Arkansas deposits are found available, St. Louis seems to be well adapted to the manufacture of fertilizers, both on account of railway facilities, its location with reference to the users of fertilizers, and also on account of the cheapness and convenience of sulphuric acid at that place.

The Map.—The map accompanying this paper is taken from the map sheets forming portions of the Manganese and Marble reports of the Geological Survey of Arkansas. It does not show, however, the entire region of the phosphate-horizon in northern Arkansas. This horizon extends northward into Missouri and westward through Boone, Carroll, Washington, and Benton counties. The horizon at which the phosphate-rocks occur further west is shown on the geological maps accompanying the Arkansas Survey's reports on Washington county (Vol. IV., *Ann. Rep.* for 1888), on Benton county (Vol. II., *Ann. Rep.* for 1891), and on those of the marble report already mentioned. These maps also have the advantage of being on a larger scale—two miles to the inch.

Historical.—The first concentrating-plant erected in the Butte district was ordered by Mr. A. Wartenweiler for the Montana Copper Co., and began operations in 1881. It was a small plant, having a capacity of only 25 tons per day, and consisted of 1 crusher, 2 pairs of rolls, for fine and coarse crushing, 4 trommels, 4 Harz jigs and 2 elevators. Frue vanners were afterward put in for the treatment of the slimes. The capacity of the mill was increased the following year to 100 tons per day, and under the present ownership (the Boston and Montana Consolidated Copper and Silver Mining Company) the vanners were replaced by a 20-foot double round table, with a pitch of 1 inch to 1 foot. It is claimed by the management that this double round table does as much and as good work as 6 or 8 vanners. An analysis of the Colusa ore, which was first concentrated by this mill, is not at hand: but it is reported that the copper was nearly all in the form of a sulphide or glance, with small quantities of chalcopyrite and bornite, associated with pyrite and a quartz gangue. The silver contents were too small in value to pay for the separation from the matte produced by smelting the concentrates.

The next concentrator in the district was put up by the Parrot Silver and Copper Co. in April, 1882. It was equipped with a Blake crusher, three-compartment Harz jigs and Frue vanners, its daily capacity being about 50 tons. The following year this was increased to 100 tons by the addition of jigs for handling fine sand which had formerly gone to the vanners; and in 1884 the mill was enlarged to 200 tons daily duty, and Rittinger tables were introduced. The facilities of the plant were still further increased in 1888 by the addition of machinery for crushing and treating middlings, and Frue vanners took the place of the Rittinger tables. At present the average work of the mill is 200 tons of ore per day, of which the following is an approximate analysis: copper, 10; iron, 15.8; sulphur, 20.5; silica, 50.1 per cent. From this a product is obtained of about one-half the weight of the crude ore and of the following composition: copper, 17; iron, 27; sulphur, 35; silica, 18 per cent. Of the 200 tons treated daily the jigs handle 177 tons, or 88.5 per cent., and the balance, 23 tons of slimes, goes to the tanks which supply the vanners and Rittinger tables. The composition of these slimes is as follows:

copper, 7.3; iron, 6.6; sulphur, 9.6; silica, 74.6 per cent. Only about 65 per cent. of these slimes goes to the vanners and tables, 35 per cent. being lost entirely in the overflow from the above-mentioned tanks. This overflow contains about 1200 pounds of copper daily, or 3 per cent. of the copper contained in the crude ore. The vanner-product is about 4 tons out of the 15 or 16 tons handled, and these concentrates carry 25 per cent. of copper. The tailings from the jigs contain 1.5 per cent. of copper, or about 2640 pounds, for the day's run. Figures for the total loss are very incomplete. It is the opinion of Mr. James Breen, the chemist of the works, that it does not exceed 10 per cent.; but in view of the loss of over 3 per cent. in overflow-slimes, this estimate of the total percentage is undoubtedly too low. Regarding the character of the ore, Mr. Breen says it is a peacock-copper, containing: copper, 53.2; iron, 17.4; sulphur, 28.5 per cent.; corresponding very closely with the formula Cu_3FeS_3, or bornite. This is mixed in a quartz gangue with pyrite, assaying: copper, 3.8; iron, 44.3; sulphur, 50.3 per cent. In the first-class ore, which is sent directly to the smelter without concentrating, the ratio of the bornite to the pyrite is 30 to 70, while in the second-class ore it is 36 to 64—that is, the first-class contains about 6 per cent. more iron in proportion to the copper than the second-class. Just below the limit of surface-decomposition in the Parrot vein the ore contains a little glance, and it is very noticeable that the percentage of the pyrite is much greater in the lower than in the upper workings, though the copper-contents remain about the same.

In the fall of 1881 work was begun by the Colorado Smelting and Mining Company on a concentrating-plant designed to have a daily capacity of 50 tons. It was completed in March, 1882, and contained the usual equipment of a breaker, Cornish rolls and sizing-screens, 7 four-compartment Harz jigs and 2 single-deck round tables. These tables were 14 feet in diameter, with a pitch of an inch to the foot, and were set to make $1\frac{1}{4}$ revolutions per minute. The work done by these tables was so imperfect that one of them was taken out in 1886, and 3 Frue vanners were substituted. Experiments were made with a Hooper slimer, and the work done by this machine was compared with that of the round table and of the Frue vanners.

The contest resulted in a decided victory for the Frue vanners. The following figures show the tests on Gagnon ore, the character of which will be described hereafter:

	Crude.		Concentrates.		Tailings.
	Silver. Ounces per ton.	Gangue. Per cent.	Silver. Ounces per ton.	Gangue. Per cent.	Silver. Ounces per ton.
Frue, . . .	$16\frac{3}{4}$	74	51	24	$2\frac{1}{2}$
Hooper, . . .	$17\frac{1}{2}$	73	43	35	14
Round Table, . .	17	74	45	33	11

In the Hooper slimer concentration was effected in 80 iron troughs having corrugated bottoms. The troughs or sluices were 63 inches long and $4\frac{3}{4}$ inches wide, with a slight inclination away from the feed, and were attached to a slowly-moving endless chain. As the sluices passed over the end of the machine, jets of water washed out the contents. Here a classification was attempted, the mineral in the upper 24 inches being clean concentrates, the next 18 inches middlings (which went through the process again) and the lower 21 inches tailings. To facilitate the separation of the sulphides from the gangue the whole frame carrying the chain was given a shaking motion by a very ingenious contrivance, which varied the stroke from one-half to three-quarters of an inch, and the number of strokes from 184 to 234. It had been claimed for the slimer that it would do as much work as four Frue vanners; but it was found in the competitive test that the vanner was handling 3168 pounds of material per 24 hours as against 3312 pounds for the slimer. The slimer was first employed in the treatment of graphite at Lake Champlain, where the crushed material was passed through the sluices and the light graphite washed away and collected in settling-tanks, while the heavier quartz and other gangue-minerals remained in the sluices, and were disposed of as the sluices passed over the end of the machine.

The round table and Hooper slimer were condemned and replaced with Frue vanners; and at about the same time a Huntington mill and two four-compartment Harz jigs were put in to treat the tailings from the coarse jigs. By working over these coarse tailings, and allowing no material coarser than $\frac{1}{8}$-inch in diameter to leave the mill in the tailings-sluice, an addi-

tional saving in concentrates of about 7 per cent. of the contents of the crude ore was effected. After these changes, the plant had an average capacity of 85 tons per day; 65 tons per day on Gagnon silver-copper ore, with a hard quartz gangue, or 100 tons on copper-silver ores like those of the Parrot and Speculator. The amount of water required was 300 gallons per minute, or about 5000 gallons per ton of ore.

The business of the Colorado Smelting and Mining Company in the purchase of silver- and silver-copper ores, and the production of a silver-copper matte, required a concentrating-plant to handle ores of this character which were too siliceous to be smelted without previous dressing; and the work of its mill has been almost entirely upon such ores, though it has also had something to do with copper- and copper-silver ores. Its principal duty has been on the ores of the Gagnon vein, the property of the company.

Gagnon Ore.—Volume XVI. of the *Transactions* contains an interesting paper by Mr. Richard Pearce on the association of minerals in the Gagnon vein, and some of the results stated below are from his personal study and investigation. During the last year the concentrator treated 16,395.7 tons of ore from this vein, which averaged 16.2 ounces of silver, and about 55 per cent. of gangue, yielding a product of 6,738.56 tons, assaying 31.1 ounces of silver and 17 per cent. of gangue. This showed that 78.8 per cent. of the silver contained in the crude ore was recovered in concentrates.

Mr. Pearce made a vanning-test on a sample of the Gagnon ore, making analyses of both crude and concentrates, and obtaining the following interesting results:

	Crude. Per cent.	Concentrates. Per cent.	Saved. Per cent.
Gangue,	49.90	15.30	
Pb,	.33	.57	78.83
Fe,	10.45	19.63	85.68
Zn,	15.08	23.60	71.36
Cu,	2.70	4.60	77.69
S,	19.14	31.83	75.84
	97.60	95.53	

Mr. Pearce draws these conclusions from the above analyses:

"In all probability the following minerals go to make up the ore:

	Crude. Per cent.	Concentrates. Per cent.	Saved. Per cent.
Gangue, SiO_2, etc., . .	49.90	15.30	
Galena, PbS, ...	.38	.65	79.50
Pyrite, FeS_2, ...	20.70	39.17	86.29
Zinc-blende, ZnS, . .	22.62	35.40	71.42
Bornite, $3(Cu_2S)$, Fe_2S_3, .	4.85	8.26	77.40
	98.45	98.78	

"The zinc-blende has sustained the greatest loss, and this mineral must have been very low in silver. The only mineral in which there is any doubt is bornite. I have figured all the copper into that mineral, and in all probability a portion of it should belong to copper pyrites (Cu_2S, Fe_2S_3)."

The above vanning-tests may be compared with the actual work in the mill on this lot, which gave the following figures: Crude ore, 350.5 tons, assaying 14.9 ounces silver, 2.8 per cent. (wet) copper and 49 per cent. gangue. Concentrates, 170.187 tons, assaying 25.4 ounces silver, 4.3 per cent. (wet) copper and 15 per cent. gangue. Tailings, by difference, 179.313 tons, assaying 4¼ ounces silver. This shows a yield of 1 ton of concentrates out of 2.06 tons of crude ore, and a recovery of 83 per cent. of the silver and 74.5 per cent. of the copper. The tailings contained 14.6 per cent. of the silver, the discrepancy being due to errors in sampling.

Parrot Ore.—On a lot of Parrot ore recently treated, which was concentrated 2.47 into 1, the following results were secured; Crude, 1976 tons, assaying 10.2 ounces silver, 0.16 ounce gold per ton, 12.4 per cent. copper and 55.5 per cent. gangue. Concentrates, 798.372 tons, assaying 20.75 ounces silver, 0.215 ounce gold per ton, 27.5 per cent. copper and 9.5 per cent. gangue. Tailings, 1,177.628 tons, assaying 2.4 ounces silver and 1.8 per cent. copper. Percentage recovered in concentrates: silver, 82.2; copper, 89.6; gold, 54.3. Percentage lost in tailings: silver, 14; copper, 8.7. Percentage unaccounted for: silver, 3.8; copper 1.7. Lower tailings and a higher percentage of saving could have been obtained by not attempting to get the concentrates so clean, but there would have been a greater weight on which to pay smelting-charges, and clean work was more profitable. The overflow from the tanks which supplied the vanners ran into a large tank, the settlings in which were kept separate on this run. No effort was made to concentrate this stuff, as no machine could handle it to advantage. The accumulation in

the tank for the entire run of 1976 tons was 11 tons, assaying 9.2 ounces silver and 10.5 per cent. copper. This accounted for 0.5 per cent. of the silver and 0.47 per cent. of the copper listed above as unaccounted for; but the remainder, 3.3 per cent. of the silver, and 1.23 per cent. of the copper, may have gone off in the final overflow, though errors in sampling may explain a portion of it.

Manganese-Ores.—Argentiferous manganese-ores have been treated by the Colorado Smelting and Mining Company's concentrator in large quantities. The oxidized surface-ores of this class show a heavy loss, except where the silver is largely associated with a hard pyrolusite, and not with the earthy forms of manganese oxides. Thousands of tons of rhodochrosite ores, containing 40 to 50 per cent. of silica, have been worked successfully, though the separation of the manganese minerals from quartz, where there is a difference of only one unit in the specific gravity, is not easy or complete. The silver in these ores is associated with blende, galenite, pyrite, and tetrahedrite. The Burlington mine has been an important source of such ores, and the results of concentration of several thousand tons may be interesting. The crude ore assayed about 12 ounces silver and .025 ounce gold, and contained about 43 per cent. of silica and 45 per cent. of carbonate of manganese. The degree of concentration was 2.2 tons into 1, and the product assayed 21 ounces silver, 0.03 ounce gold, with 18 per cent. of gangue, while the tailings assayed 5 ounces silver. The concentrates contained about 77 per cent. of the silver-contents of the crude ore. An analysis by Mr. O. Bergström of one lot of concentrates gave the following results:

Analysis.

	Per cent.
SiO_2,	14.90
Al_2O_3,	1.55
S,	6.50
Fe,	6.30
MnO,	38.89
Zn,	4.00
Cu,	.30
Pb,	Trace
CO_2,	25.60
Combined water,	1.80
	99.84

Mineral Composition.

	Per cent.
ZnS,	6.00
Cu_2S,	.45
FeS_2,	8.14
FeO, CO_2,	5.19
MnO, CO_2,	61.70
MnO,	.80
Al_2O_3,	1.55
SiO_2,	14.90
H_2O,	1.80
	100.53

It is the opinion of Mr. Pearce that some rhodonite (MnO,SiO_2) exists in the ore, and that some of the SiO_2 is combined in that way, and he further suggests that the Al_2O_3 exists in combination with the SiO_2 and H_2O in the form of kaolinite or as partially kaolinized feldspar. The concentrated product from this rhodochrosite-ore was smelted in reverberatory matte-furnaces without previous calcining, and the manganese made it a desirable flux.

Steam-Stamps and Cornish Rolls.—While the cost of crushing ore is much less under a steam-stamp than through crushers and Cornish rolls, the percentage of slimes is much greater from the stamp than from the older method of crushing, and the most recently constructed mills in Butte do not contain steam-stamps. Samples of Gagnon ore treated by the two methods were sent to Mr. Henry A. Vezin, and were sifted through screens of varying mesh with the following results: The screen used with the steam-stamp was perforated steel, with $\frac{5}{16}$-inch round holes on front and back and $\frac{3}{8}$-inch on sides.

Sample of Gagnon Ore, Through Blake Crusher, Cornish Rolls and Mesh-Screen.

Meshes per linear inch.	Size in millimeters.	Weight. Oz. avoir.	Per cent.	Silver-assay. Oz. per ton.
4 to 6	4 to 2.8	7.55	9.03	12.7
6 to 10	2.8 to 2	29.	34.70	14.3
10 to 14	2 to 1.4	13.55	16.21	13.9
14 to 20	1.4 to 1	9.78	11.70	14.1
20 to 30	1 to 0.6	6.20	7.42	15.3
30 to 80	0.6 to 0.16	9.56	11.44	17.5
80	0.16	7.94	9.50	23.6
		83.58	100.00	

Sample of Gagnon Ore from Steam-Stamp, Through "Perforated Metal" Screen.

Meshes per linear inch.	Size in millimeters.	Weight. Oz. av.	Per cent.	Assay. Silver. Oz. per ton.	Assay. Copper, per cent.	Assay. Gangue, per cent.
4 to 6	4 to 2.8	2.95	2.40	8.	2.3	67.2
6 to 10	2.8 to 2	10.10	8.22	9.9	2.5	62.4
10 to 14	2 to 1.4	10.10	8.22	10.2	2.7	61.4
14 to 20	1.4 to 1	11.20	9.12	10.23	2.6	58.8
20 to 30	1 to 0.6	10.87	8.85	9.93	2.3	57.3
30 to 40	0.6 to 0.42	9.55	7.77	10.8	2.5	53
40 to 60	0.42 to 0.25	11.86	9.66	11.9	2.7	49.1
60 to 120	0.25 to 0.12	7.67	6.24	14.1	3.1	46.7
120	0.12	48.55	39.52	18.9	4.3	46.8
		122.85	100.00			

It will be seen from these figures that in the steam-stamp work 63 per cent. of the ore was fine enough to pass through a screen having 30 meshes to the inch, and in the other system only 21 per cent. would pass through the same size of screen. At the Anaconda works the ore passes over grizzlies, and thus all ore which is fine enough for treatment on the jigs and tables does not pass under the steam-stamp; and in modern Butte mills, where rolls are used, ores which come from the mines in a condition fine enough for concentration (*i.e.*, the "mine fines") do not pass through the rolls. By this separation of the fine ore before crushing the percentage of slimes is greatly reduced. Given abundant space for settling and a sufficient number of concentrating-machines doing close work on the slimes there is an argument in favor of fine crushing, for we find that grains of quartz from the jigs no larger than $\frac{1}{8}$ of an inch in diameter carry away, attached to or locked up in them, a large percentage of the value of the ore. There would be less objection to the amount of the slimes if the ores from the steam-stamp were classified through screens instead of hydraulic separators, which, by the necessary addition of water, greatly increase the volume of material to be handled or settled.

Water-Requirements.—There is a great difference in the quantity of water used in the several Butte mills for the treatment of a ton of ore. In the concentrators where steam-stamps and hydraulic separators are in use the amount of water required is not less than 5000 gallons for each ton of ore, while in mills of the other class 3000 gallons will do the work.

Part II.

The first part of this paper was written in 1889, but the publication was delayed until descriptions could be obtained of several Butte concentrating-plants. Prof. R. H. Richards suggested the form which is used in the following descriptions. This method has an advantage over the graphic form of describing mill-processes, as it avoids the folded sheets, which are not convenient in the Institute publications.

The Anaconda Mill, Anaconda, Montana.

In 1883 the Anaconda Mining Co., under the direction of Manager Marcus Daly, began the construction of a concentrating-plant at Anaconda, 28 miles by rail from the mine. It had a capacity of about 1000 tons per day, and commenced operations in October, 1884. No provision was made at first for the concentration of slimes, which were settled in tanks and sent to the smelter, but later on the Frue, Triumph and Embrey vanners were introduced; and the mill as completed may be described as follows:

1. Ore-bins, receiving ore from mines in railroad-cars; thence to (2).
2. Grizzlies; over-size to (3); under-size to (4).
3. Twelve Blake crushers; thence to (4).
4. Hoppers or bins; thence in cars, weighed and sampled, to (5).
5. Tulloch automatic feeders; thence to (6).
6. Six pairs of 15-by-27-inch rolls; thence to (7).
7. Six trommels, 3-mesh; over-size to (8); under-size to (9).
8. Six pairs of 15-by-27-inch rolls; thence back to (7).
9. Six trommels, 5-mesh; over-size to (10); under-size to (11).
10. Twelve 4-compartment Harz jigs, 4-mesh; concentrates to (21); tailings to waste.
11. Six trommels, 6-mesh; over-size to (12); under-size to (13).
12. Twelve 4-compartment Harz jigs, 5-mesh; concentrates to (21); tailings to waste.
13. Six trommels, 8-mesh; over-size to (14); under-size to (15).

14. Twelve 4-compartment Harz jigs, 6-mesh; concentrates to (21); tailings to waste.

15. Six trommels, 10-mesh; over-size to (16); under-size to (17).

16. Twelve 4-compartment Harz jigs, 8-mesh; concentrates to (21); tailings to waste.

17. Six hydraulic sizers; coarse to (18); overflow or slimes to (19).

18. Twelve 4-compartment Harz jigs, 10- and 12-mesh; concentrates to (21); tailings to waste.

19. Twelve settling-tanks; settlings to (20); overflow to waste.

20. Twenty-six Frue vanners, 10 Triumph and 10 Embrey vanners; concentrates to (21); tailings to waste.

21. Draining-floor; thence in cars to roasting-furnaces.

In 1886 Mr. Daly placed the works in charge of Mr. Otto Stahlmann, and experiments were made, having in view the reduction of cost of treatment. The results proved so satisfactory that all the old machinery was thrown out, and steam-stamps, Collom jigs, hydraulic separators and circular slime-tables were put in. The capacity of the plant was thus increased to 2000 tons per day.

About the same time Mr. Stahlmann designed a new mill, which was built at Carroll, 2 miles from the old works, has a daily capacity of 1750 tons, and may be described as follows:

Anaconda Concentrator, Carroll, Montana.

1. Seven receiving-bins; thence to (2).

2. Seven grizzlies, 1¼-inch bars, 1-inch spaces; coarse to (4); fine to (3).

3. Seven trommels, $\frac{3}{16}$-inch round holes, perforated steel; coarse to (3*b*); fine to (26).

3*b*. Seven elevators; thence to (4).

4. Seven steam-stamps (15 by 30-inch cylinders, 90 strokes per minute) screen $\frac{3}{16}$ by ⅜-inch; thence to (5).

5. Twenty-eight hydraulic separators: 1st spigot to (6); 2d to (7); 3d to (8); 4th to (9); overflow to (22*a*).

6. Twenty-eight Collom jigs, 4- and 8-mesh: 1st sieve-discharge to (37); 2d to (37); 1st sieve-hutch to (37); 2d to (37); tailings to (10).

7. Twenty-eight Collom jigs, 8- and 10-mesh: 1st sieve-hutch to (37); 2d to (37); tailings to (10).

8. Twenty-eight Collom jigs, 10- and 12-mesh: 1st sieve-hutch to (37); 2d to (16); tailings to waste.

9. Twenty-eight Collom jigs, 12- and 14-mesh: 1st sieve-hutch to (37); 2d to (16); tailings to waste.

10. Fourteen settling-boxes: spigots to (11); overflow used on (17) and (18).

11. Fourteen circular jigs, 8-mesh: discharge to (12); hutch to (16); tailings to waste.

12. Fourteen Heberle grinders; through 10-mesh to (13).

13. Fourteen hydraulic separators: 1st spigot to (14); 2d to (15); overflow to (22*a*).

14. Fourteen No. 1 grinder jigs, 10- and 12-mesh: 1st hutch to (37); 2d to (31); tailings to waste.

15. Fourteen No. 2 grinder jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (34); tailings to waste.

16. Fourteen hydraulic separators: 1st spigot to (17); 2d to (18); overflow to (22*a*).

17. Fourteen No. 1 finishing-jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (19); tailings to waste.

18. Fourteen No. 2 finishing-jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (19); tailings to waste.

19. Four hydraulic separators: 1st spigot to (20); 2d to (21); overflow to waste.

20. Four No. 1 final finishing-jigs, 14- and 14-mesh: 1st hutch to (37); 2d to (37); tailings to waste.

21. Four No. 2 final finishing-jigs, 14- and 14-mesh; 1st hutch to (37); 2d to (37); tailings to waste.

22*a*. Settling tanks: settlings to (23); overflow with concentrates.

22*b*. Settling tanks: settlings to (24); overflow with concentrates.

22*c*. Settling tanks: settlings to (25); overflow with concentrates.

23. Forty-eight slime-tables, 2 on a shaft, 21 feet diameter, $\frac{1}{8}$-inch pitch: heads to (37); tailings to waste; middlings to (22*c*).

24. Twenty slime-tables, 1 on a shaft, 21 feet diameter, $\frac{1}{4}$-inch pitch: heads to (37); tailings to waste; middlings to (22*c*).

25. Twenty-four slime-tables, 1 on a shaft, 21 feet diameter, $\frac{1}{4}$-inch pitch: heads to (37); tailings to waste.

26. Twelve hydraulic separators: 1st spigot to (27); 2d to (28); 3d to (29); 4th to (30); overflow to (22*b*).

27. Twelve Collom jigs, 4- and 8-mesh: 1st sieve-discharge to (37); 2d to (37); 1st hutch to (37); 2d to (37); tailings to waste.

28. Twelve Collom jigs, 8- and 10-mesh: 1st sieve-discharge to (37); 2d to (37); 1st hutch to (37); 2d to (37); tailings to waste.

29. Twelve Collom jigs, 10- and 12-mesh: 1st sieve-hutch to (37); 2d to (34); tailings to waste.

30. Twelve Collom jigs, 12- and 14-mesh: 1st sieve-hutch to (37); 2d to (34); tailings to waste.

31. Six hydraulic separators: 1st spigot to (32); 2d to (33); overflow to settling-tanks.

32. Six finishing-jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (19); tailings to waste.

33. Six finishing-jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (19); tailings to waste.

34. Six hydraulic separators: 1st spigot to (35); 2d to (36); overflow to settling-tanks.

35. Six finishing-jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (19); tailings to waste.

36. Six finishing-jigs, 12- and 14-mesh: 1st hutch to (37); 2d to (19); tailings to waste.

37. Great concentrates-launder to (38).

38. First concentrates tank-house: 8 elevated tanks, 12 by 150 feet, with discharge-gates, and tracks and cars to smelter; overflow to (39).

39. Second concentrates tank-house: 5 cement tanks on the ground, 40 by 80 feet, with floor-tracks for cleaning up and shipping sediment to smelter; overflow to (40*a*).

40*a*. Settling-pond, or lake: sediment to smelter; overflow to (40*b*).

40*b*. Settling-pond, or lake: sediment to smelter; overflow to (40*c*).

40*c*. Settling-pond, or lake: sediment to smelter; overflow to waste.

NOTE.—In the foregoing description, by "hutch" is meant the concentrates which pass through the jig-screens, and by "discharge," the material which is taken off the screens either on the sides or ends.

Since the above notes were taken changes have been made, among which are the following:

Ore which is notably soft is crushed by a steam-stamp having screens with ½-inch round holes, and 1-sieve Harz jigs are used, the tailings from which are crushed by a steam-stamp through ¼-inch round-hole screens.

Collom jigs have been substituted for the circular jigs.

Cornish rolls are now used instead of Heberle grinders.

The Butte Reduction-Works, Butte, Montana.

(Daily capacity, 200 tons.)

1. Two-ore bins, 100 tons capacity: thence to (2).
2. Grizzlies, bars ¾-inch apart: oversize to (3); undersize to (4).
3. Steam-stamp, 95 drops per minute, screens $\frac{7}{16}$-inch round holes; thence to (6).
4. Trommel, $\frac{7}{16}$-inch round holes: oversize to (5); under size to (6).
5. Harz jig, one-compartment, 2 by 3½ feet, screens 4-mesh, No. 12 wire, 3½-inch stroke: hutch to (11); discharge by conveyor to (33); tailings to (3).
6. Four hydraulic sizers: 1st spigot to (7); 2d to (8); 3d to (9); 4th to (10); overflow to (21).
7. Four Collom jigs, 2 sieves, 6- and 8-mesh; 1st sieve-discharge to (30); hutch to (30); 2d sieve-discharge to (11); hutch to (30); tailings to waste.
8. Four Collom jigs, 2 sieves, 10- and 10-mesh: 1st sieve-discharge to (30); hutch to (30); 2d sieve-discharge to (11); hutch to (19); tailings to waste.
9. Four Collom jigs, 2 sieves, 10- and 10-mesh: 1st sieve-hutch to (30); 2d to (20); tailings to waste.
10. Four Collom jigs, 2 sieves, 14- and 14-mesh: 1st sieve-hutch to (30); 2d to (20); tailings or overflow to (27).
11. One pair of 16-by-30-inch steel rolls; thence to (12).
12. Elevator; thence to (13).
13. Trommel (3 mm.): oversize to (11); undersize to (14).
14. Trommel (2½ mm.): oversize to (15); undersize to (16).
15. Harz jig, 3 compartments, 8-, 8- and 10-mesh: 1st sieve-discharge and hutch to (30); 2d to (30); 3d sieve-discharge to (11); 3d hutch to (19); tailings to waste.

16. Settling-tank: 2 spigots to (17); overflow to (15) and (18).

17. Hydraulic sizer: spigot to (18); overflow to (6).

18. Harz jig, 3 compartments, 8-, 10- and 10-mesh: 1st sieve-hutch to (30); 2d to (30); 3d to (19); tailings to waste.

19. Two Collom jigs, 2 sieves, 10- and 14-mesh: 1st sieve-hutch to (30); discharge to calciners; 2d sieve-hutch to (12); tailings to waste.

20. Two Collom jigs, 2 sieves, 14- and 14-mesh: 1st sieve-hutch to (30); 2d to (12).

21. Settling-tank: settlings by elevator to (22); overflow to (34).

22. Sizing-tank: 1st spigot to (23); 2d to (24); 3d to (25); 4th to (26).

23. Double-deck Clark's improved riffled table: upper-deck concentrates to (30); middlings to lower-deck; tailings to waste. Lower-deck concentrates to (32); middlings to (28); tailings to waste.

24. Double-deck plane-table, diameter 18 feet, pitch 1⅛ inches to 1 foot. Same treatment as on Clark's table, except that middlings from lower-deck go to settling-tanks before going to vanners (29).

25. Double-deck Clark table, same as (23).

26. Double-deck plane-table, same as (24).

27. One Frue vanner: concentrates to calciners; tailings to waste.

28. One Frue vanner: concentrates to calciners; tailings to waste.

29. One Frue vanner: concentrates to calciners; tailings to waste.

30. Elevator; thence to small bin; settlings to (31); overflow to (5).

31. Ten storage-bins, 300 tons: overflow to tanks for settling fine mineral; drainings to (30).

32. Tank for settling slimes; thence to calcining-furnaces.

33. Storage- and drainage-bin; thence to roasting-stalls.

34. Large settling-ponds: settlings to smelter: overflow to waste.

NOTE.—Alterations now being made in the mill-arrangements will affect the classification of the stamp-discharge through trommels down to 2½-millimeter size; and the Evans excentric-jigs are being substituted for the Collom machines.

The Parrot Concentrator of the Parrot Silver and Copper Company, Butte, Montana.

The following is a description of the process in 1893, the capacity being 300 tons. The mill was built originally for 150 tons, and no middlings were saved.

1. Ore-bin, thence to (2).

2. Blake crusher, 9 by 15 inches; thence to (3).

3. Elevator No. 1; thence to (4).

4. Trommel No. 1 ($\frac{5}{8}$-inch holes): oversize to (5); undersize to (9).

5. Blake crusher, 7 by 10 inches; thence to (6).

6. Rolls No. 1 (15 by 30 inches); thence to (7).

7. Rolls No. 2 (same size); thence to (8).

8. Horizontal carrying-belt (14 inches); back to (3).

9. Trommel No. 2 ($\frac{3}{8}$-inch holes): oversize to (10); undersize to (11).

10. Jig No. 1 (in duplicate), 2 compartments, 4-mesh; all concentrates: discharge to (51); overflow to (26).

11. Trommel No. 3 ($\frac{1}{4}$-inch holes): oversize to (12); undersize to (13).

12. Jig No. 2 (in duplicate), 3 compartments, 3-mesh; all concentrates: hutch to (51); overflow to (26).

13. Trommel No. 4 ($\frac{1}{8}$-inch holes): oversize to (14); undersize to (15).

14. Jig No. 3 (in duplicate), 3 compartments, 5-mesh; all concentrates: hutch to (51); overflow to (30).

15. Hydraulic sizer, 3 divisions: drawn off to (16), (17), (18), (19), and (20); overflow from 3d division to (21).

16. Jig No. 4 (in duplicate), 5-mesh, supplied from 1st division (15), 2 compartments concentrates, 3d compartment middlings: hutch to (51); middlings to (30); waste overflow to (53).

17. Jig No. 5 (in duplicate), supplied from 2d division (15); 2 compartments concentrates, 5-mesh; 3d compartment, middlings, 8-mesh: hutch to (51); middlings to (30); waste overflow to (53).

18. Jig No. 6 (in duplicate), supplied from 3d division (15), 4 compartments, 8-mesh; last compartment, middlings, which are shoveled back: hutch to (51); waste overflow to (53).

19. Jig No. 7 (in duplicate), supplied from 3d division (15),

4 compartments, 10-mesh; last compartment, middlings, which are shoveled back: hutch to (51); waste overflow to (53).

20. Jig No. 8 (in duplicate), supplied from 3d division (15), 4 compartments, 14-mesh; last compartment, middlings, which are shoveled back: hutch to (51); waste overflow to (53).

21. Five spitzlutten: drawn off to (22); overflow to (23).

22. Ten vanners (5 Frue, 3 5-foot Tulloch, and 2 4-foot Tulloch): concentrates shoveled out; waste overflow to (53).

23. Large settling-tank for de-watering: drawn off to (24); overflow to (25).

24. Four vanners (2 Frue and 2 Embrey): concentrates shoveled out; waste overflow to (53).

25. Outside settling-tanks: settlings raw into reverberatories.

26. Launder No. 1; thence to (27).

27. Elevator No. 2; thence to (28).

28. Rolls No 3 (15 by 30 inches); thence to (29).

29. Elevator No. 3; back to (9).

30. Launder No. 2; thence to (31).

31. Elevator No. 4; thence to (32).

32. Rolls No. 4 (15 by 30 inches); thence to (33).

33. Trommel No. 5 ($\frac{1}{8}$-inch holes): oversize to (34); undersize to (35).

34. Rolls No. 5 (15 by 30 inches); thence also to (35).

35. Elevator No. 5; thence to (36).

36. First hydraulic sizer of middlings: drawn off to (37), (38), and (39); overflow to (44).

37. Middlings-jig No. 1 (in duplicate); 2 compartments, concentrates, 3d, middlings, 6-mesh: hutch to (51); middlings to (40); waste overflow to (53).

38. Jig No. 2 (in duplicate): same as (37), and same disposition.

39. Jig No. 3 (in duplicate): 8-mesh, same as (37), and same disposition.

40. Launder No. 3; thence to (41).

41. Elevator No. 6; thence to (42).

42. Launder No. 4; thence to (43).

43 Rolls No. 6 (15 by 30 inches); back to (33).

44. Second hydraulic sizer: drawn off to (45), (46), (47), and (48); overflow to (49).

45. Jig No. 4 (in duplicate), 2 compartments, concentrates,

3d, middlings, 8-mesh: hutch to (51); middlings to (35); waste overflow to (53).

46. Jig No. 5 (in duplicate), 10-mesh, same as (45): same disposition.

47. Jig No. 6 (in duplicate), 3 compartments, concentrates, 4th, middlings, 14-mesh: hutch to (51); middlings to (35); waste overflow to (53).

48. Jig No. 7 (in duplicate), 2 compartments, concentrates, 3d, middlings, 14-mesh: hutch to (51); middlings to (35); waste overflow to (53).

49. Two spitzlutten: drawn off to (50); overflow to (25).

50. Two Embrey vanners: concentrates shoveled out; waste overflow to (53).

51. Launder No. 5, for concentrates; thence to (52).

52. Elevator No. 7, for concentrates; to trommel ($\frac{1}{4}$-inch holes): oversize, raw to blast-furnace; water with undersize to storage-bins.

53. Launder No. 6, for tailings: thence to (54).

54. Three elevators, Nos. 8, 9 and 10; to tailings-dump by launder.

NOTE.—In place of (4), a 6-foot trommel with $\frac{5}{8}$-inch holes, there has been substituted a trommel 9 feet in length with $\frac{7}{8}$-inch holes for the last 3 feet.

The jigs handling the coarse material are fitted with an improved jig-excentric, the invention of Mr. Hermann A. Keller.

This company's new plant at Whitehall, Montana, will be arranged in two divisions with a total capacity of 700 tons.

Practically the same system will be used: crushing through rock-breakers and rolls, followed by trommels as far as practicable, that is, to $\frac{3}{32}$-inch. Below that size hydraulic separators will be employed, the sand going to fine jigs, and the slimes to vanners.

The Butte and Boston Concentrator, Butte, Montana.

(Daily capacity, 600 tons.)

The following is a description of the practice in 1894:

1. Two ore-bins, 200 tons capacity each; thence to (2).
2. Two 9-by-15-inch Blake crushers; thence to (3A).
3. Two trommels.

A. 1st half, 30 by 48 inches, $1\frac{1}{2}$-inch mesh: oversize to (3B); undersize to (7A).

B. 2d half, 30 by 48 inches, $2\frac{1}{8}$-inch mesh; oversize to (5); undersize to (4).

4. Two 1-compartment jigs, speed 140 revolutions, stroke $3\frac{1}{2}$ to 5 inches; hutch to (6); discharge (from top of screen only) to (32); tailings to (5).

5. Two 4 by 10-inch Blake crushers; thence to (6).

6. Two elevators, speed 350 feet per minute, 10 by 5 by 6-inch buckets, 12 inches apart; thence to (7A).

7. Two trommels.

A. 1st half, $15\frac{1}{2}$ mm.; oversize to (7B); undersize to (11).

B. 2d half, $1\frac{1}{2}$ inches; oversize to (10); undersize to (8).

8. Two elevators; thence to (9).

9. Two 1-compartment jigs, speed 140 revolutions, stroke $2\frac{1}{2}$ to 4 inches; hutch to (11); discharge (from top of screen only) to (33); tailings to (10).

10. Two pairs of rolls, 16 by 30 inches, speed 38 revolutions; thence to (6).

11. Elevator, same size and speed as (6); thence to (12).

12. Four trommels, 36 by 60 inches, $8\frac{1}{2}$ mm.; oversize to (13); undersize to (15).

13. Four 1-compartment jigs; hutch to (11); discharge to (29); tailings to (14).

14. Two pairs of rolls, 15 by 26 inches; speed 40 revolutions; thence to (11).

15. Four trommels, 36 by 60 inches, $4\frac{1}{2}$ mm.; oversize to (16); undersize to (17).

16. Four 3-compartment jigs; hutch to (29); discharge to (29); tailings to (23).

17. Four 2-compartment hydraulic separators; 1st spigot to (18); 2d spigot to (19); overflow to (20).

18. Four 3-compartment jigs; hutch to (29); discharge to (29); tailings to (23).

19. Four 3-compartment jigs; hutch to (29); discharge to (29); tailings to (23).

20. Two 2-compartment hydraulic separators; 1st spigot to (21); 2d spigot to (22); overflow to (26).

21. Four 4-compartment jigs; hutch to (29); no discharge; tailings to (23).

22. Four 4-compartment jigs; hutch to (29); no discharge; tailings to (23).

23. Four Huntington mills, 5-feet diameter, speed 70 revolutions, 12½ mm. by 2½ mm.; thence to (24).

24. Two 1-compartment hydraulic separators; spigot to (25); overflow to (26).

25. Six 4-compartment jigs; hutch of first three sieves to (31); hutch of last sieve to (28); tailings to waste.

26. Four V-settling-tanks, 16 feet long, 69 inches deep; settlings to (27); overflow, if any, to waste.

27. Twenty Frue vanners, 4 feet wide; concentrates to (31); tailings to waste.

28. Elevator; thence to (25).

29. Double elevator (two 10-inch belts, 400 feet per minute, 10-inch buckets, 12 inches apart); thence to (30).

30. Four bins for storing and draining concentrates (200 tons each); thence to calciners.

31. Elevator (10-inch belt, 400 feet per minute, 10-inch buckets, 12 inches apart); thence to (29).

32. Two storage-bins; thence to calciners.

33. Two storage-bins; thence to calciners.

The Colorado Concentrator of the Colorado Smelting and Mining Company, Butte, Montana.

(Daily capacity, 300 tons.)

The following is a description of the process:

1. Ore-bin, about 300 tons capacity; thence to (2).

2. Blake crusher, 9 by 15 inches; thence to (3).

3. Elevator (in duplicate), 14-inch belt; thence to (4).

4. Trommel (in duplicate), 20 mm., ½-inch perforated steel; oversize to (5); undersize to (6).

5. Two Blake crushers, 7 by 10 inches; thence to (7).

6. Trommel (in duplicate), 7 mm., ⅛-inch perforated steel; oversize to (6*a*); undersize to (9).

6*a*. Two jigs, 1 sieve each, 4-mesh, stroke 2-inch, speed 140 revolutions; hutch by sluice to (17); end-discharge to (6*b*); tailings to (8).

6*b*. Rolls, 14 by 27 inches; thence to (25).

7. Two sets of rolls, 16 by 30 inches; thence to (3).

8. One set of rolls, 16 by 30 inches; thence to (3).

9. Trommel (in duplicate), $4\frac{1}{2}$ mm., $\frac{3}{32}$-inch perforated steel; oversize to (10); undersize to (11).

10. Two jigs, 3 compartments, 3-mesh, 190 strokes; hutch to (25); discharge to (25): tailings to (17).

11. Trommel (in duplicate), 3 mm., $\frac{1}{16}$-inch perforated steel; oversize to (12); undersize to (13).

12. Four jigs, 3 compartments, 6-mesh, 210 strokes; hutch to (25); discharge to (25); tailings to (17).

13. Two hydraulic separators, 4 compartments; settlings of all divisions to (14); overflow to (15).

14. Six jigs, 3 compartments, 10-mesh, 230 strokes; hutch to (25); end-discharge to (17); side-discharge to (25); tailings (waste) to (24).

15. V-tank; settlings to (16); no overflow.

16. Sixteen Frue vanners: concentrates to (25); tailings to waste.

17. Unwatering trommel, $2\frac{1}{2}$ mm.: oversize to (19); undersize, with water, to (18).

18. Settling-tank: settlings to (19); overflow to (24).

19. Rolls, 16 by 30 inches; thence to (20).

20. Elevator, 12-inch belt; thence to (21).

21. Trommel, $2\frac{1}{2}$ mm.: oversize back to (19); undersize to (22).

22. Hydraulic separator, 1 compartment: settlings to (23); overflow to (15).

23. Four jigs, 3 compartments, 10-mesh: hutch to (25); end-discharge to (17); side-discharge to (25); tailings (waste) to (24).

24. Elevator, 20-inch belt; thence through sluices or launder to tailings-dump.

25. Elevator, 12-inch belt; thence to (26).

26. Thirteen storage-bins, 100 tons each: concentrates in cars to smelter, overflow and drainage to settling-tanks.

Scheme for Getting Additional Saving from Jig-Tailings.

(Rejected for reasons stated on page 628.)

Tailings from (14) and (23) to (27).

27. Settling-tank: settlings to (28); overflow to (24).

28. Elevator, 12-inch belt, 10-by-7-inch buckets, $36\frac{1}{2}$ feet between shafts; thence to (29).

29. Two trommels, 1 mm.: oversize to (30); undersize to (31).

30. Rolls, 16 by 30 inches, 76 revolutions; thence to (28).

31. Settling-tank: settlings to vanners.

The new Colorado concentrator was finished in the spring of 1892, and as it is the most recently constructed concentrating-plant in the Butte district, its machinery and equipment are described more particularly, as follows:

The power-arrangements include three 54-inch by 16-feet boilers, which are fed by a 3-plunger pump, made by Wm. Baragwanath & Sons, Chicago, and a Korting injector is used whenever repairs are necessary on the pump. The fuel is "slack" from the Rock Springs coal-mines in Wyoming, which costs $3.50 per ton in Butte. Its quality is shown by the following analysis:

	Per cent.
Moisture,	4.00
Volatile,	37.00
Carbon,	52.40
Ash,	6.60
	100.00

A tandem compound condensing Corliss engine, with high-pressure cylinder 14 inches in diameter, low-pressure 24 inches and 42 inches stroke, furnishes the power. The water used in condensing passes from the surface condenser under a pressure from the tank at the top of the mill-building sufficient to return it to another tank in the mill, which supplies the water required for the jigs and vanners. The condensed water is used for boiler-feed and is raised in temperature from 75 to 135 degrees by passing through a heater, through which the exhaust of the engine passes on its way to the condenser. The indicated horse-power of the engine, when the mill is treating the hardest ores, at the rate of 218 tons per day day, is 177, while on the softer ores it has shown as low as 171 horse-power for 376 tons per day. These figures will convey an idea of the variations in cost of concentrating different ores of the district.

Besides running the entire mill-machinery, as described, the engine drives an 80-kilowatt (100 horse-power) generator, made

by the C. & C. Electric Motor Co. of New York, running 640 revolutions and developing 500 volts, 62 amperes. The current is carried 2800 feet through a No. 2 wire to a 65 horse-power C. & C. electric motor, running 600 revolutions and belted to a counter-shaft, from which belts drive the two ends of a Roots rotary pump, No. 5, throwing 10 gallons per revolution to a height of 90 feet and through 2800 feet of 10-inch wooden pipe. This pipe was made by boring out logs and banding them spirally with 1-inch No. 22 iron. It is covered with a mixture of asphalt and sawdust. In explanation of the use of wooden pipe, it should be said that the corrosive mine-waters of Butte are discharged into the stream which furnishes the supply for the mill, and iron pipe would not long withstand the action of the acids.

The amount of water required for the concentrator is about 600 gallons per minute, and with this quantity there is no overflow from the vanner feed-tanks, or, in other words, no untreated material leaves the mill—every particle of ore which enters the mill is given a chance to be saved either by jigs or vanners.

Life of Roots' Rotary Pump.—As numerous inquiries have been made regarding this machine, some facts in its history are here given. It was made by the P. H. & F. M. Roots Co., Connersville, Ind., and embodies the principles of the Roots blower, manufactured by the same company.

When the pump above mentioned was started it would throw 10 gallons of water per revolution, under a head, as shown by indicator, of 46 pounds, and in driving it 60 revolutions per minute the horse-power required, as calculated from volts and amperes, was 27½. After being in service 672 days, and having pumped about 600,000,000 gallons, the pump was so badly worn that it would only throw 3.9 gallons per revolution, and the horse-power required to drive it 115 revolutions per minute was 51. The "case," "pistons" and shafts were then replaced by new ones, at a cost of $872.00, and the case was sent back to the manufacturer, to be bored out, and have new pistons fitted to it, after which it was returned and held in reserve. It should be added that the life of the pump was much shortened by acid water and by the silt contained in it during the spring months.

Appliances for Crushing Tailings or Middlings.

In the Butte mills we find Cornish rolls, Huntington mills, Heberle grinders and stamps in use for crushing tailings and middlings; but we are only able to compare Huntington mills with steel rolls for doing this work.

At the Butte and Boston, where Huntington mills are employed, the cost is 8.8 cents per ton of tailings treated, not including the cost of power. Good judgment, however, is required to get satisfactory results from these mills. If a concentrating-plant is to be equipped with them, it is necessary: first, since repairs have to be made frequently, to put in at least one more mill than is required for the actual amount of material to be crushed; and secondly, to be sure that ample capacity is provided. Any overcrowding of the mill causes the shoes or rolls to slip when they should roll against the ring or die; and the result is that the circular form of the rolls is soon lost and the efficiency reduced. Moreover, it must be borne in mind that the action of the machine has a tendency to drive the material against its periphery, and hence the grains strike the screens, in passing, at a very acute angle. If it has been determined that no grains of mineral or quartz of larger diameter than $\frac{1}{10}$-inch should leave the mill, it is not necessary to put on screens having $\frac{1}{10}$-inch holes; $\frac{1}{8}$-inch screens will obtain the desired result. The screens in the Huntington mills of the Butte and Boston Co. have slots, $2\frac{1}{2}$ by $12\frac{1}{2}$ millimeters in diameter, with the longest dimension horizontal.

At the Colorado concentrator, where Cornish rolls with steel shells are employed, the cost per ton of tailings treated is 4.6 cents, including all expenses for screens, elevators, rolls, shells, etc., but not including power. The relative cost of power in the Huntington mills and the Cornish rolls has not been determined; but the difference, if any, could hardly amount to 4.2 cents per ton in favor of the mills, and, therefore, the figures given are strongly in favor of rolls for economical crushing.

Losses in Jig-Tailings.

In ores like the Gagnon, where the quartz gangue is impregnated with silver- and copper-bearing minerals, it was to be expected that jig-tailings would carry off a high percentage

of the values of the original ore, even after crushing and re-treating the coarser jig-tailings and middlings, and allowing no grains of quartz larger than ⅛-inch in diameter to leave the mill; but the difference between the jig-tailings and vanner-tailings was so marked that efforts have been made at the Colorado concentrator to recover a part of the mineral values which were escaping from the jigs. The difference between jig- and vanner-tailings, as shown by the average for one month, is as follows:

	Silver, oz. per ton.	Copper, per cent.
Jig-tailings, July, 1892, ...	2.86	1.64
Vanner-tailings, July, 1892, ...	1.6	1.00

Samples of these jig-tailings were sent to Mr. Richard Pearce for tests on a vanning-shovel, and the results of two tests, after crushing the material fine enough to pass through a 50-mesh screen, were as follows:

		Concentrates.			Tailings.		Saved in Concentrates.	
		Silver.	Copper.	Gangue.	Silver.	Copper.	Silver.	Copper.
Test No.	Tons into 1.	Ounces per ton.	Per cent.	Per cent.	Ounces per ton.	Per cent.	Per cent.	Per cent.
1.	9.523	14.6	8.9	36.4	1.3	.83	53.6	57.0
2.	11.583	18.0	10.8	27.4			56.84	54.31

These figures showed that if the jig-tailings of the mill, which amounted to 123 tons per day, could be crushed fine enough for treatment on vanners, about 12 tons of concentrates would be added to the product, which would sell to the smelters for $6.00 or $8.00 per ton. On a basis of 10 tons into 1 this crushing and vanning would therefore have to be done for less than 60 cents per ton of jig-tailings in order to yield a profit to the mine. Before putting in any machinery for the treatment of the material a sample was sifted through screens of different sizes and the several siftings were assayed. First, a 40-mesh screen was used, and the following are the results of this division:

		Assay.	
	Per cent.	Silver. Oz. per ton.	Copper. Per cent.
On 40-mesh screen, ...	69.83	2.10	1.32
Through 40-mesh screen, . . .	30.17	4.10	2.01
	100.00		

This showed that material containing 45½ per cent. of the silver and 39½ per cent. of the copper lost in jig-tailings was already fine enough for treatment on vanners without further crushing, and we may remark in passing that it also shows the imperfections of the hydraulic sizing, for this 40-mesh material should have gone on with the slimes instead of going to the jigs, but having gone to the jigs it was floated off with the tailings. The results of a further division by sifting will be of interest.

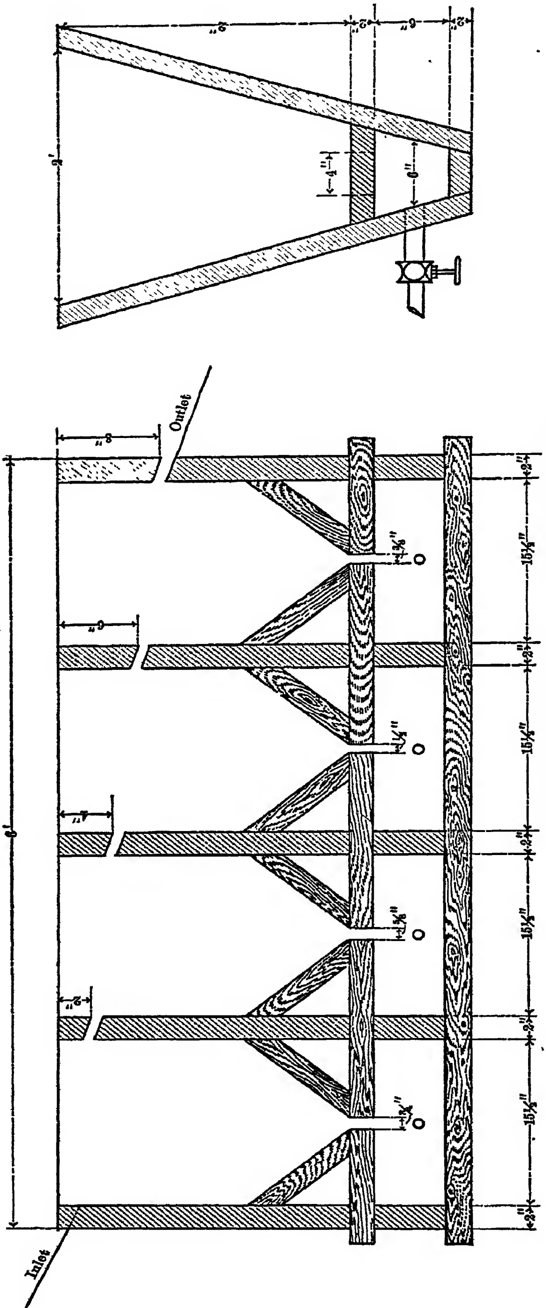
Assay before sifting: silver, 2.7 oz. per ton; copper, 1.65 per cent.

Screen.		Per cent.	Assay. Silver. Oz. per ton.	Assay. Copper. Per cent.
On 10-mesh,		3.9	3.0	1.5
Through 10 on 20,		12.65	2.5	1.2
" 20 on 40,		45.80	2.2	1.37
" 40 on 60,		18.90	2.4	1.55
" 60 on 80,		10.00	2.4	1.72
	37.65			
" 80 on 100,		4.05	3.0	1.85
" 100,		4.70	7.0	4.00

The problem to be solved may therefore be summed up as follows: Given 123 tons per day of jig-tailings, containing about 2¾ ozs. of silver per ton and 1.65 per cent. of copper, and carried along with 500,000 gallons of water, or about 17 tons of water to 1 ton of tailings, to eliminate the water and that portion of the material (amounting to about 40 tons), which is already fine enough for vanner-treatment, settle these slimes and run them over vanners, crush the sand and work this stuff also on slime-machines. Appliances for this purpose were introduced at the Colorado concentrator, and the scheme is shown in the description of that plant on a preceding page.

It was not intended to provide enough machinery for the treatment of the entire amount of tailings, but to experiment with a part of it first. To separate the portion which was already fine enough for treatment on vanners from the coarse grains, hydraulic sizing was tried with unsatisfactory results; perforated steel trommels were used with 1-mm. round holes, but the screens soon became "blind," and all efforts to keep the holes clear, either by jets of water or by automatic hammering or jarring, proved impracticable.

Fig. 1.



HYDRAULIC CLASSIFIER OF THE COLORADO S. & M. CO.

Scale, ¼"=1'

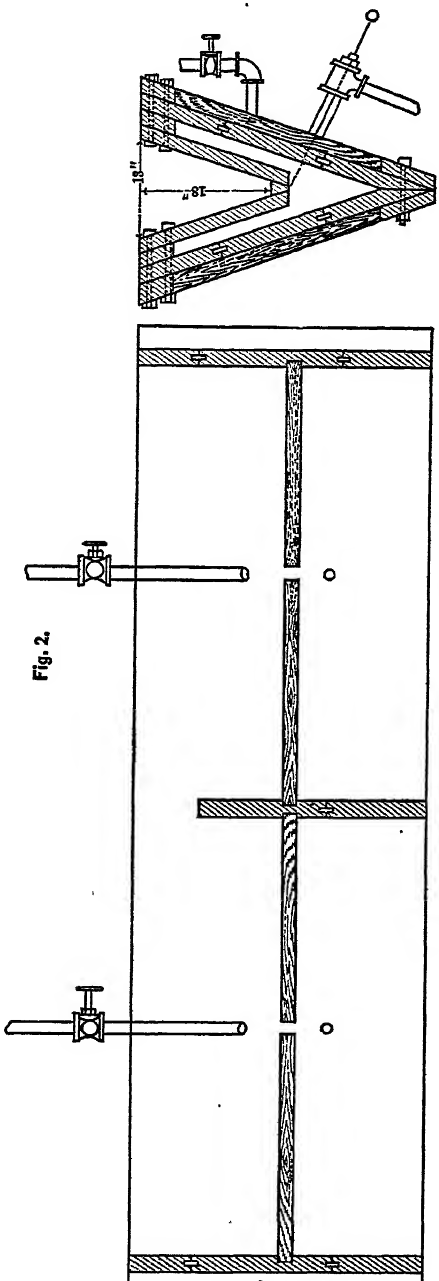


Fig. 2.

HYDRAULIC SEPARATOR OF THE BUTTE & BOSTON MINING CO.
Scale, 1/8" = 1'

The experiment demonstrated that the crushing could be done with rolls, and that if the 40 tons of slime material, with the excess of water, could be eliminated the remainder of the jig-tailings could be crushed to the fineness required for vanner-work by two sets of rolls, but the separation of the water and slimes remains an unsolved problem.

The form of hydraulic sizer from which the above sifting-tests were made is shown in Fig. 1.

In the belief that one sizer of this kind was overcrowded, another one of the same form was used, and the work was divided between the two, but the percentage of material passing through a 40-mesh screen remained about the same—with the disadvantage of having increased the volume of material to be settled in the vanner feed-tanks.

Then an Evans classifier, as described by Prof. Monroe in *Trans.*, xxii., 648, was substituted for one of these sizers, and sifting-tests of the spigot-discharges from the two different sizers were compared. The separation was no better with the Evans than with the other form of classifier.

Investigating still further the imperfections of hydraulic sizing, a sifting-test was made on jig-tailings from the Butte and Boston mill when running on ore from the Gray Rock mine. Another form of hydraulic separator was used (see Fig. 2), but the ore was of the silver-copper class, like that from the Gagnon.

		Per cent.	Assay. Silver. Oz. per ton.	Assay. Copper. Per cent.
Sample of Gray Rock tailings, . .			2.85	1.41
On 10-mesh, ...		2.26	2.60	.86
Through 10 on 20-mesh, . . .		17.56	2.60	1.21
" 20 on 40 " . . .		46.06	2.36	1.13
" 40 on 60 " . . .	34.11	16.36	2.28	1.13
" 60 on 80 " . . .		8.37	2.00	1.16
" 80 on 100 " . . .		4.16	2.00	1.35
" 100 " . . .		5.22	3.44	2.68
		99.99		

It will be seen that 34.11 per cent. of the jig-tailings should have passed on from the hydraulic sizer to the vanners.

From the following results it will be seen that the losses in jig-tailings are also considerable on the copper-silver ores:

Silver Bow.

		Per cent.	Assay. Silver. Oz. per ton.	Assay. Copper. Per cent.
Original samples,			1.32	1.07
On 10-mesh,		4.84	.96	.81
Through 10 on 20,		18.07	.96	.83
" 20 " 40,		37.08	1.08	.98
" 40 " 60,	39.95	17.24	1.20	.99
" 60 " 80,		10.23	1.12	.93
" 80 " 100,		5.59	1.28	1.07
" 100,		6.89	2.08	1.98
		99.94		

Parrot.

		Per cent.	Assay. Silver. Oz. per ton.	Assay. Copper. Per cent.
Original samples,			.80	1.05
On 10-mesh,		11.04	.68	.98
Through 10 on 20,		23.72	.84	.84
" 20 " 40,		36.64	.70	.75
" 40 " 60,	28.57	12.67	.32	.69
" 60 " 80,		7.48	.66	.87
" 80 " 100,		3.90	1.20	1.00
" 100,		4.52	1.87	1.50
		99.97		

Percentage through 40-mesh screen: Of Silver Bow tailings, 39.95.
" " " " Of Parrot tailings, 28.57.

Mr. Richard Pearce made a vanning-test on a sample of Silver Bow jig-tailings, and the following results show that this material is worthy of further treatment, if the obstacles above described can be removed.

The sample assayed 1.1 ounces silver and 1.32 per cent. copper, and it was found that 19.42 per cent. of it would pass through a 40-mesh screen; but, preparatory to the vanning-test, the whole sample was worked through a 60-mesh screen. On the vanning-shovel 100 grammes gave 10.650 grammes of concentrates assaying 7.4 ounces silver and 9.17 per cent. copper and containing 15.7 per cent. of silica or insoluble residue, showing a saving of 71.64 per cent. of the silver and 73.98 per cent. of the copper. The tailings from the shovel assayed 0.35 per cent. copper; silver not determined.

Extending our inquiries outside of Butte, we find that in the

treatment of the silver-lead ores of the Cœur d'Alene region in Idaho considerable losses are sustained in jig-tailings. The following results of vanning-tests made by Mr. Pearce on this material, from one of the mills in that district, will be of interest. The equipment of the mill includes rock-breakers, crushing-rolls, hydraulic separators, Collom jigs and circular slime-tables.

Idaho Jig-Tailings.

			Assay.		
		Per cent.	Silver. Oz. per ton.	Lead. Per cent.	Zinc. Per cent.
Sample, ...			3.1	4.5	8.9
Sizing, ...					
On 20-mesh, . . .		46.40			
Through 20 on 40, . .		28.24			
" 40 " 60, . .		8.26			
" 60 " 80, . .		5.07			
	25.36		4.8	6.7	11.1
" 80 " 100, . .		2.25			
" 100, . . .		9.78			
		100.00			

A portion of the sample was first put through a 60-mesh screen, and 100 grammes gave by vanning, 5.54 grammes of concentrates (or 18.05 tons into 1), assaying 19 ounces silver, 30.4 per cent. lead and 22.3 per cent. zinc. An effort was made in this vanning to eliminate the zinc as far as practicable, and the saving of silver was only 34 and of lead 37 per cent.

It was found that 25.36 per cent. of the tailings would pass through a 40-mesh screen, showing here again the imperfection of hydraulic sizing. The assay of this fine portion was 4.8 ounces silver, 6.7 per cent. lead and 11.1 per cent. zinc, and by vanning 7.961 tons into 1 a product was obtained which assayed 16.1 ounces silver, 24.5 per cent. lead and 23.2 per cent. zinc.

With silver at 63 cents per ounce, lead at 3.20 cents per pound, a deduction of 50 cents for each unit of zinc over 10 per cent., and a freight- and smelting-charge of $25 per ton, it will be seen that neither of the above-mentioned products would yield enough to pay charges.

The ores of the Cœur d'Alene district have to stand very high charges for freight and treatment, as they cannot find a market at a distance of less than 500 miles; but the development of other ores in their vicinity and a better price for their

metallic contents may make it profitable in the future to treat the jig-tailings.

Considerable space has been given here to the discussion of the losses in jig-tailings, since the subject is one of much importance in Butte, and has not received the attention it deserves. It is not improbable that some of the companies look forward to re-working their jig-tailings when conditions are more favorable; but there are three reasons why they should be treated, if possible, as fast as they are produced.

1. It is doubtful if the expenses of working will ever be enough smaller to balance the expense of getting the material back into the mills.

2. Unless reservoirs of ample settling-capacity are provided, the finest and richest material will be carried away into the streams by the flow of water which goes out with the tailings.

3. Oxidation and leaching will rob them of some of their values, and will also leave them in such a condition that greater losses will be incurred in working.

"Copper is the cheapest thing I've got," was the remark of a prominent mine-manager of Butte, when his attention was called to losses in concentrating which could only be avoided by a considerable original outlay, followed by an increase in operating expenses. This belief may have stood in the way of careful investigations into leaks and how to prevent them.

The most modern of the Butte mills are well equipped with economical engines and other machinery of improved design, and operating expenses are probably at a minimum with present rates for labor and material; but there is room for improvement in the saving of mineral values.

Comparative Work of Frue Vanners and Circular Tables.

Gagnon ore has been treated in mills of both types.

1. Rock-breakers, Cornish rolls, trommels, Harz jigs and Frue vanners.

2. Steam-stamp, hydraulic sizing, Collom jigs and circular tables.

An improved circular table, designed by the Clark Brothers, of Butte, is manufactured by the Western Iron Works of that city. Instead of the plain surface, 11 concentric steps are made, each having a width of 8 inches and a pitch of $1\frac{1}{8}$ inches

to the foot, the pitch of the old plain-surface tables being $1\frac{1}{8}$ inches to the foot, from center to circumference. These tables have given better results than the old ones; but the work done by vanners on Gagnon ore is much more satisfactory than on either kind of circular tables, as will be seen by the following:

Gagnon Ore.

Crude assay, 12.5 ounces silver, 4.7 per cent. copper, 55 per cent. gangue.

	Vanners.			Old Tables.			Clark Tables.		
	Silver. Ozs. per ton.	Copper. Per cent.	Gangue. Per cent.	Silver. Ozs. per ton.	Copper. Per cent.	Gangue. Per cent.	Silver. Ozs. per ton.	Copper. Per cent.	Gang. Per cent.
Concentrates, .	33.5	12.3	17.0	21.5	10.4	39.1	25.5	11.5	22.4
Tailings, . .	1.5	1.0		3.95	1.6		3.55	1.4	

On the copper-silver ores of the district the difference between the two machines is not as great; but still the record is in favor of the vanners.

The above results show that the vanners give cleaner concentrates and lower tailings than the tables, and the difference in favor of the vanner is great enough to counteract the objections which are urged against it—namely, original outlay, cost of repairs and attention, and small capacity.

The first cost of the two machines in Butte, set up and ready to run, is about the same, and may be taken at $500 for each. Vanners in the Butte mills are only required to handle 2 or 3 tons per day, and it is claimed for the tables that they will treat four times as much. If this is true, the original cost of equipment for working a given quantity of ore would be four times as great for vanners as for tables. (In the gold-mills of Montana, where concentration follows amalgamation on plates, two vanners handle the ore from five stamps; hence each machine has to treat about 5 tons per day.)

In the cost of labor for attention the ratio is about the same for the two machines as in the item of first outlay. Vanners are now arranged in the Butte mills so as to discharge concentrates continuously through sluices and elevators into bins, which also receive the product of the jigs, for settling and draining, and one man on a shift can take care of 20 or 24 machines. With an unvarying pressure of water, the same number of tables can probably be watched by one man.

In repairs the vanner is undoubtedly more expensive than the table; but if it has good care this item of cost is not excessive. The average life of a good belt is five years. Adding to the price of a belt the cost of other worn-out parts during the same period, it has been found that the total outlay in repairs is about $150, or 3.3 cents per ton of ore treated by the machine itself, or only $\frac{1}{3}$ of a cent per ton of ore going into the mill (the amount of slimes being only about 10 per cent. of the total amount treated by the mill). The first injury which usually occurs to a vanner-belt is the cracking of the raised edges or flanges. This has been remedied to a great extent by new forms of flanges, among them H. G. Blasdel's patent, in which the flanges "stand at an acute angle toward the center, and therefore readily conform to the change of direction while passing over the end rollers."

In one of the mills treating Butte ores, where both tables and vanners are in use, the middlings from the former are treated on the latter. The results are very satisfactory.

Tests on Vanner-Tailings.

In discussing the "Observations Concerning Ore-Dressing," by O. Bilharz, *Trans.*, xxii., Professor Richards, on page 701, lays down the rule that "every machine, as far as practicable, should have its guard." This is certainly a wise precaution, and a guard or provision for treatment of the tailings of some concentrating-machine is almost imperative; but there are cases in which the yield would hardly pay for the effort to make an additional recovery, as will be seen from the following:

The accumulated daily samples of vanner-tailings, assaying 1.5 ounces silver and 1 per cent. copper, from a 3400-ton lot of Gagnon ore were sent to Mr. Richard Pearce for a test, to determine if it would be profitable to place a slime-table after the vanners for the treatment of the tailings, and he reported:

"Three hundred grammes gave 4.36 grammes of concentrates, which were very siliceous, and which assayed 7.3 per cent. copper and 14.7 ounces silver to the ton. The material saved was very light and required careful handling on the vanning-shovel to collect it. I cannot see how you are going to make this pay the expense of additional treatment."

This shows a recovery of only 1 ton out of 69 tons handled,

or only 1 ton in two days; and as the net yield of the product at the smelter would not be more than $10 per ton, it is evident that the cost of re-handling this material should be less than 14½ cents per ton in order to make any profit. With the certainty that no slime-table could recover as much as the vanning-shovel, the above figures did not encourage the company to provide facilities for elevating, settling and concentrating the vanner-tailings.

There are sixty-six 4-foot Frue vanners in the Butte concentrators, but other machines of the belt-type are getting a foothold in the mills of the district. One of the advantages claimed for these newer machines, in comparison with the Frue, is that they prevent the accumulation of drifts of sand at the edges of the belt. It is asserted that these drifts drag over the tailings-end of the belt some of the values which would otherwise move on to the concentrates-end.

In order to determine how much difference there was between the tailings discharged from the edges and from the middle of the belt, two samples were taken on Gagnon tailings, one from a zone 6 inches wide on each edge, the other from 3 feet in the middle, and the assays were as follows:

	Middle.		Edges.	
	Silver. Oz. per ton.	Copper. Per cent.	Silver. Oz. per ton.	Copper. Per cent.
Average of 4 samples, ...	1.61	.69	1.60	.80
" " " "	1.45	.84	2.20	1.51
Accumulated samples for 12 hours, .	1.20	1.05	2.	1.35

Sifting-tests were made on the last mentioned samples to ascertain whether the shake of the machine did not throw the larger particles of sand to the edges of the belt; for, if it were found that the tailings from the edges contained more coarse sand than from the middle, the higher assays from the former might be accounted for, in part at least, by the presence of included grains of silver- and copper-bearing minerals in these coarser sands. But the tests demonstrated that there was practically no difference in the fineness of the material from the edges and middle, and therefore the higher assays from the former can only be accounted for by the accumulation of sand on the edges which prevents the free passage of concentrates over the proper end of the machine.

Other Belt-Machines.

A brief mention of several belt-machines and their devices for preventing the accumulation of sand will not be out of place.

The Embrey vanners, four of which are in use at the Parrot concentrator, have an end-shake, and thus do away with side-drifting of sand. Their work is very satisfactory.

At the Western Iron Works, in Butte, the Tulloch concentrator is made, and two of them are in use at the Parrot, and seven at the Montana Ore Purchasing Company's mill. They have canvas belts and an oscillating motion, which is intended to imitate the movement of panning, and are giving satisfaction.

The attention of mill-men has also been called to the Woodbury concentrator, in which seven narrow belts take the place of a wide one, and to the Johnson concentrator, made by the Risdon Iron Works, of San Francisco, which has a shaking motion and several other peculiar features of apparent advantage; also to the Union concentrator, manufactured by the Union Iron Works, of San Francisco.

A belt-machine which resembles the Bilharz table in design is made by J. M. Montgomery, of Butte.

Corrugated Vanner-Belts.—Experiments with these belts gave unsatisfactory results on Butte ores. They did not yield as clean concentrates or as low tailings as the plain belts.

Gilpin County Concentrator.—In the gold-mills of Colorado a shaking-table known by the above name, with end-bump, has been extensively used in saving pyrites. On low-grade tailings, which will not stand the first cost and operating expense of vanners, this machine may do well enough, but on the Butte ores it has not been found satisfactory. A clean product could not be made except with too great a loss in the tailings.

Classification of Slimes for Vanner-Work.

In one of the Butte mills, where seven Frue vanners were in use, the V-shaped feed-tanks were arranged so as to classify the material for the vanners, and the results on one lot of silver-copper ore were as follows:

1. Launder carrying all slimes from mill to (2).
2. V-shaped feed-tank: settlings to (3); overflow to (4).
3. Four Frue vanners: tailings, 1 ounce silver; 1 per cent. copper.
4. V-shaped feed-tank: settlings to (5); overflow to (6).

5. No. 5 vanner: tailings, 0.64 ounce silver; 0.9 per cent. copper.

6. V-shaped feed-tank: settlings to (7); overflow to (8).

7. No. 6 vanner: tailings, 1.30 ounces silver; 1.2 per cent. copper.

8. V-shaped feed-tank: settlings to (9); overflow to (10).

9. No. 7 vanner: tailings, 4 ounces silver; 2.7 per cent. copper.

10. Settling-tank: settlings, 9.2 ounces silver; 7.7 per cent. copper.

The assays of tailings given above will show how difficult it is to save the fine sulphides; and it is the opinion of Mr. Pearce, verified by practical experience on the Butte ores, that better results are obtained on vanners when no classification is made—provided the material going upon the machines contains no grains of mineral too large for the vanner to handle properly. There is a reason for this in the effect which larger and heavier grains of material would have in entrapping the finer slimes and holding them down to the surface of the belt.

Applying this theory to the mixing of different kinds of ore for concentration, it has been found that better results were obtained by mixing some of the Butte ores than by working them separately.

Amalgamation of Concentrator-Tailings.

Mention has already been made (see page 599) of the silver-ores of Butte and of the unsatisfactory results of their treatment by concentration.

The Nettie mine has shipped more ore of this class to concentrating-works than any other mine in the district, and the high assay of the tailings led to a series of tests to determine what percentage of the silver was amalgamable.

A partial analysis of 16,379 tons so treated at the Colorado concentrator is as follows:

	Per cent.	
Silica,	66.50	
Manganese,	5.56	
Iron,	4.90	
Lead,	2.44	
Zinc,	4.60	
Sulphur,	6.43	
Copper,	.10	
Silver,	.08	= 23.7 ounces per ton.
Gold,		0.03 " "
	90.61	

The product, assaying 45 ounces silver, amounted to 6371 tons, and the degree of concentration was, therefore, 2.57 tons into 1. The average assay of the tailings was 9 ounces of silver.

Percentage saved in concentrates, ...	73.8
" lost in tailings, ...	23.2
	97.0

Preparatory to an amalgamation-test on the tailings, samples were sifted through sieves of different fineness, and the results were as follows:*

	Nettie Jig-Tailings. Per cent.	Nettie Jig-Tailings. Silver Assay. Ozs. per ton.	Nettie Vanner-Tailings. Per cent.	Nettie Vanner-Tailings. Silver Assay. Ozs. per ton.
Sample sizing, ...		10.		12.
On 10-mesh, ...	4.90	10.1	1.04	14.2
Through 10 on 20, . .	18.82	9.4	1.44	10.4
" 20 " 40, . .	47.12	9.9	5.13	7.4
" 40 " 60, . .	15.58	8.9	13.33	6.3
" 60 " 80, . .	6.77	8.8	10.87	6.0
" 80 " 100, . .	2.59	9.3	12.44	7.0
" 100, ...	4.17	14.1	55.72	13.6
	99.95		99.97	

A quantity sufficient for several pan-charges was sent to the Lexington Company's mill for test-runs through the amalgamating-pans, and Mr. C. C. Rueger reports the following results:

Charge No.		Assays. Pan Charges.	Assays. Settler Tailings.	Percentages. Lost in Tailings.	Percentages. Saved (by Difference).
	Vanner-tailings:	Ozs.	Ozs.		
1	Amalgamation without grinding......No chemicals..............	11.6	9.2	79.3	20.7
2	" with " ...... " "	11.4	9.0	78.9	21.1
3	" " " ... 4 pounds blue-stone. No salt.	10.8	6.2	57.4	42.6
4	" " " ... 8 pounds blue-stone. 20 " salt.	11.4	4.4	38.6	61.4
5	" " " ... 8 " blue-stone. 30 " salt.	11.6	4.0	34.5	65.5
6	" " " ... 12 " blue-stone. 30 " salt.	12.0	3.6	30.0	70.0
	Jig-tailings:				
7	Amalgamation, etc........................ 8 " blue-stone. 30 " salt.	10.0	6.4	64.0	36.0
8 9	" " .... 12 " blue-stone. 30 " salt.	10.0	4.2	42.0	58.0

The weight of pan-charges was about 3000 pounds, and the time 8 hours.

* In collecting the vanner-tailings for this test they were allowed to flow into a pool, and a certain amount of concentration took place, giving settlings much richer in silver than the average tailings from the mill.

The above sifting-test shows that 92.36 per cent. of the vanner-tailings were fine enough to pass through a 40-mesh sieve before amalgamation, and the settler-tailings showed from 94.74 to 95.20 per cent. through a 40-mesh. Of the jig-tailings, before going into the pans, 29.11 per cent. passed through 40-mesh, and, after grinding in the pans, 41.96 per cent. passed through the same sieve.

Taking the average assay of the concentrator-tailings at 9 ounces, with silver at 66 cents per ounce, and assuming that the pans would recover two-thirds of the value, the yield per ton would be only about $4.

Cost of Concentrating.

Managers of Butte mills are unwilling to publish figures of the cost of treatment; but it may be stated in a general way that the expense ranges from 35 cents per ton in the larger plants to $1 in those of smaller capacity. The ores of the district differ so greatly in character that these figures have no particular significance, and where custom-work is done, which is the case in the smaller mills, the frequent changes from one lot of ore to another make the costs higher than those of steady work. In mills which have the advantage of water-power it is not improbable that the cost is less than 35 cents.

If we assume that the average assay of the concentrating-ores of Butte is 7 per cent. copper (wet assay), and allow for all losses in concentrating and smelting, we may calculate that, in the concentrating-department alone, each pound of copper obtained in the matte has cost one-third of a cent.

Roll-Shells.

One important item of expense is in castings for the crushing-rolls, and many experiments have been made with different kinds of metal, having in view the reduction of this cost. Chilled-iron shells, furnished by the local foundries, have been tried frequently; but as the problem of getting iron-castings which are both hard and tough has not been solved, steel is in more common use. At the Colorado concentrator, where rolls have the most severe duty, cast-steel from several manufacturers has been tried, and the cost per ton of ore for shells of this

metal has been about 4¾ cents. Tests have also been made of the wearing qualities of "rolled-" chrome- and manganese-steels. The latter, which is said to contain 12 per cent. of manganese, has given better satisfaction than any other, not only for roll-shells, but also for crusher-plates. It is made by the Taylor Iron and Steel Co. of High Bridge, New Jersey. One pair of manganese-steel roll-shells remained in service while 52,000 tons were treated by the mill, and a pair of ordinary cast-steel on the same duty wore out while 24,000 tons were treated. The manganese-steel rolls weighed 2651 pounds, and, although they wore down to thinner scrap than the cast-steel, the waste metal weighed 650 pounds, representing 24.5 per cent. of the original castings. As there is no market for this or other steel-scrap at local foundries, the waste is quite an important matter.

Composition of Butte Ores.

The following partial analyses will show what the ores of the district contain:

	No. 1.*		No. 2.		No. 3.			No. 4.†	
	Smelting.	Concentrating.	Smelting.	Concentrating.	A. Smelting.	B. Smelting.	Concentrating.	Smelting.	Concentrating.
Cu	20.26	6.50	9.80	4.60	29.95	26.25	8.10	.55	.10
Fe	14.42	9.55	26.00	12.10	18.65	16.40	10.10	5.10	4.90
Zn	5.30	8.66	......	......	7.25	2.34	......	16.29	4.60
S	26.34	15.87	33.60	15.70	36.15	29.18	14.88	13.19	6.43
Mn	.45	.09	......	......	.26	......	......	6.73	5.56
As	1.66	.94	......	......	.44	4.00	......	trace	......
Sb	trace	.20	......	......	......	......	......	.05	......
Pb	trace	.25	......	......	......	......	......	8.55	2.44
Al_2O_3	2.35	......	......	......	......	......	......	.....	......
(Insol. Res.)	28.72	54.80	28.70	65.20	7.26	21.50	63.00	43.22	66.50
	99.50	96.86	......	......	99.96	99.67	96.08	93.68	90.53
Ag	30 ozs.	10 ozs.	11.2ozs	3.3 ozs.	24 ozs.	20 ozs.	6.5 ozs.	43 ozs.	20 ozs.
Au	$0.44	$0.44	No assay.	No assay.	No assay.	No assay.	No assay.	$0.80	$0.40

* Gagnon.

It will be noticed that the gold-assay does not increase with the silver and copper contents, the smelting ore being no better in gold than the concentrating class.

Hübnerite (tungstate of manganese) has been found, but not frequently, in the Gagnon ore. In the analysis WO_3 was looked for, but not found in appreciable quantity.

† Nettie.

Magnetic Observations in Geological Mapping.

BY HENRY LLOYD SMYTH, CAMBRIDGE, MASS.

(Colorado Meeting, September, 1896.)

SECTION I.—INTRODUCTION.

IN 1891–92 I was entrusted with the geological survey of part of the large area lying between the Marquette and Menominee iron-ranges in the Upper Peninsula of Michigan, and extending from the Republic trough on the north, through portions of the valleys of the Michigamme and Fence rivers, to and including the intermediate Felch mountain range on the south.

Until a few years back the larger part of this area has been very difficult of access, and much of it is difficult still. The rock-surface is for the most part covered with swamp, forest and glacial deposits of various kinds, and therefore has not invited geological study. For these reasons the district, as a whole, with the exception of the Felch mountain range, remained almost unknown from a geological standpoint until our work in 1892.

Our main object was to map the pre-Cambrian rocks, particularly the iron-bearing formations of Algonkian or Huronian age, and to obtain as much information as possible with reference to their sequence and structure. In carrying out this purpose, the use of simple magnetic instruments was found to be of the greatest assistance, and indeed indispensable. In this paper the results of our experience in tracing magnetic rocks by means of the disturbances produced in these instruments, have been brought together in systematic form, with such references to the particular area as are necessary in order to illustrate general principles, with the hope that they will prove useful to other workers in the same region, or in other regions where similar conditions may be encountered.

The writer was efficiently aided in the field-work by Messrs. Samuel Sanford and Charles N. Fairchild for nearly the whole period, and E. B. Mathews and H. F. Phillips for part of it,

as assistant geologists, and by Messrs. Lewis and Forbes as skilled woodsmen.

As has been said already, the area in which our work was done is largely drift-covered, to somewhat varying but usually considerable depths; the mantle on the whole is so evenly spread, that outcrops of any rocks, except those belonging to the Archæan, are, in many sections, few and scattered, and sometimes are almost entirely lacking over whole townships.

Under these circumstances, and in view of the additional fact that the pre-Cambrian rock-structure is likely to be complex, even a general outlining of the old formations would be impossible by the usual geological methods, and if we were restricted to these there would be no alternative but to map most of the territory as Pleistocene. It happens, however, that the Algonkian rocks of Michigan contain a large amount of magnetite, which is known from observation in the developed ranges to be characteristic of certain geological formations. It undoubtedly occurs in more or less amount in all the sedimentary rocks, and is also present, sometimes in considerable quantities, in rocks that are not sedimentary, as, for example, round the margins of the old intrusive diorite bosses. But generally speaking, its occurrence in large quantities is confined so closely to definite geological formations, in which it is found in characteristic association with certain other minerals, or to horizons within those formations, that it can be guardedly used in identifying them, and in tracing them from localities where they outcrop through areas in which they are buried. This use is not only justified, from an empirical standpoint, by the presumption in favor of analogies to which no exceptions are known, but it has a rational basis, in the view of the late Professor Irving,* which is steadily gaining ground, that at least much of the iron of this magnetite was originally buried in the same formations in which it now occurs, through the agency of organic life. From this point of view the magnetite is in a certain sense a fossil, but with the important practical advantage over other organic remains, that it need not be dug up in order to prove its existence.

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* "Classification of the Early Cambrian and Pre-Cambrian Formations," *Seventh Annual Report of the U. S. Geological Survey*, pp. 451–52.

These magnetite-bearing rocks always produce disturbances in the compass-needles held in their neighborhood. By a systematic location and comparison of these disturbances, the position of the rocks which produce them can be determined with a considerable degree of precision, even when they are deeply buried. Besides their position on the map, the magnetic observations may, and often do, indicate certain other geologically-important facts, such as whether the rocks are flat-lying or highly-tilted, the direction of strike and dip, and in some cases, the depth to which they are buried. The methods employed in the field-work were based on those described by Major T. B. Brooks,* who perfected the dial-compass, and predicted the importance of magnetic methods in geological mapping; but the results reached in interpretation were gradually developed in the progress of this work, as we were daily brought face to face with phenomena which called for explanation.

It must be clearly understood at the outset that in the iron-ranges of the south shore of Lake Superior magnetite is rarely concentrated in large bodies, and that, in fact, its known occurrence as such is restricted to a small part of the western Marquette district, where in one producing mine it now forms practically the whole product, and in another a variable, but usually important part of the whole. It is, therefore, well understood in the Upper Peninsula that disturbances of the magnetic needle, however great, do not mean workable deposits of magnetite. Whatever significance such disturbances possess is stratigraphical, and properly interpreted may lead to discoveries of rich ore, other than magnetite, in formations to the position and attitude of which the attractions may furnish a clue. But it may be asserted as a general proposition, the essential truth of which has been established by the experience of many years, that in the region referred to magnetic disturbances usually mean that magnetic iron-ore in a workable deposit does not exist in the area of disturbance.

Section II.—Description of the Magnetic Rocks.

The magnetic rock of the Lower Huronian series of the western portion of the Marquette area (which is of special impor-

* *Geological Survey of Michigan*, vol. i., Part I., Chapter VII.

tance to notice, since it forms one of the chief horizons of reference to which our work is tied) is the Negaunee iron-formation. It is finely exposed at the south end of the Republic trough; but farther north has been greatly reduced in thickness, or locally cut out altogether,* by the upper Marquette denudation, and, when present at all, is usually drift-covered.

It is made up essentially of the two minerals, quartz and grünerite (which, previous to Dr. Lane's work,† had almost always been assumed to be actinolite), and either or both of the two anhydrous oxides of iron, magnetite and hematite. It was shown in the same paper‡ that, of these two, magnetite was characteristic rather of the lower portion and hematite of the upper portion of the formation. The total thickness is very unequally divided between the two iron-minerals; the magnetic portion on Republic mountain occupying more than five-sixths of the whole, while a short distance north, in proportion as the formation has been more deeply cut into from above by the Upper Huronian denudation, the red jasper, or that variety in which hematite predominates, dwindles often to disappearance.

This rock often possesses a very distinct banding, which preserves, as we believe, the original planes of sedimentation, since it is parallel to the bounding surfaces of the formation. This banding is caused by the alternation of layers, in which one of the constituent minerals predominates over the others, sometimes, indeed, to their total exclusion. In the lower part of the formation, quartz and grünerite constitute the bulk of the rock, with magnetite scattered somewhat indiscriminately through them. Higher up, the magnetite and quartz relatively increase, until near the top, but below the jasper, the grünerite goes out almost entirely, and the rock consists of quartz-bands, heavily charged with magnetite, in alternation with bands of nearly pure magnetite. In the Negaunee formation, as exposed at Republic, the magnetite therefore occurs concentrated in some of the parallel bands and disseminated through the others.

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* *15th Ann. Rep. U. S. Geological Survey*, pp. 613, 617.

† *Am. Jour. Sci.*, vol. xlii., 1891, p. 505.

‡ *15th Ann. Rep. U. S. Geological Survey*, pp. 611, 612.

Any precise or even rough quantitative determination of the distribution of the magnetite from point to point, both across and along the strike, is of course impossible; but the study of the exposures, and particularly the long series of magnetic observations, lead to the belief that it is on the whole remarkably uniform.

In the same district there is another much less prominent locus of magnetite at and near the base of the upper Marquette or "hanging-wall" quartzite. This marks an old and now crumpled surface of denudation, as well as certain layers near the bottom of the conglomerate that was deposited upon it. Along the strike of this zone, which is of small thickness, the distribution of magnetite is very irregular; and for this and the additional reason that when the magnetic portion of the Negaunee formation comes up to it the disturbances which it produces cannot be discriminated from those produced by the latter, the position of the plane usually cannot be inferred. But it is important from an economic standpoint, since it is the home of the only known rich bodies, and, therefore, heavy and localized disturbances which can be referred to it are always of possible significance.

Higher up in the Upper Huronian series a magnetic rock having much the same characters as the Negaunee formation was encountered in the western part of the Marquette district, on the northern border of the area covered by our work. This rock was not found again in outcrops, nor is there any reason to believe, from the magnetic work, that it occurs elsewhere in this area.

In the Menominee district and its extensions there are also two horizons in the lower series, characterized by the presence of magnetite. The lower of these is not known to outcrop, but it occurs somewhere near the junction of the dolomite and the underlying quartzite. From hand-specimens collected at a test-pit in the Felch mountain range, its lithological affinities would seem to connect it more closely with the quartzite than with the dolomite. The magnetic disturbances due to this formation are feeble, but they are quite persistent in the Felch mountain area, and have thrown some light on the geological structure.

The other formation which produces disturbances is that

which I have correlated with the Negaunee formation and named in a former paper* the Michigamme jasper.

This rock, while varying a good deal in character, is generally much like that magnetic phase of the Negaunee formation in which the grünerite is rare or absent. From the fact that the jasper now survives for the most part only in shallow and shattered synclines, it often lacks the regular banding; and hematite is always present in greater or less amount. The relative proportions of the two iron minerals vary along the strike also. The rock as a whole, however, is very magnetic, but not so strongly so as the Negaunee formation in the Republic trough.

In the Felch mountain range there is still a third magnetic formation, which seems to overlie unconformably the lower series, and is therefore provisionally assigned to the Upper Huronian. This formation consists of ferruginous schists, interstratified with layers of ferruginous fragmental quartzite. It is generally much less highly inclined than the magnetic rocks of the lower series as well as less rich in iron, and the disturbances produced by it are correspondingly small.

Besides these rocks of sedimentary origin, with which this paper properly deals, it may be mentioned that along the Fence river there is a considerable area of metamorphic eruptives, which are often exceedingly magnetic. These are restricted to a definite geological horizon, within which the magnetic disturbances are remarkable for their complexity and irregularity, no doubt as the result of a very irregular distribution of magnetite, and of the formations which chiefly contain it. The rocks in this belt outcrop freely, and the disturbances can therefore easily be assigned to the proper causes.

SECTION III.—THE DISTRIBUTION OF MAGNETISM IN THE MAGNETIC ROCKS.

Magnetite occurs therefore in these Algonkian rocks in different ways. In some instances it is mainly concentrated in nearly pure parallel layers; in others, it is more or less evenly disseminated through non-magnetic material; and still again, it is present in both forms. Moreover, these rocks have all

* *Am. Jour. Sci.*, vol. xlvii., 1894, pp. 217, 218, 223.

been folded, more or less strongly, at more than one period; and wherever they are exposed they are seen to be inclined to the horizon, often at high angles, and to be traversed by intersecting sets of joint- and cleavage-planes, some of which always cut the bedding, and often have been the seat of the development of secondary minerals. By the crossing of these various surfaces, the rocks are divided up into small unit-masses, at the boundaries of which there is either an actual physical parting, or a break in the continuity of the magnetite.

It is well known that when a bar-magnet is broken, and the severed ends are again joined, the two pieces do not unite to form one magnet, but remain as two. It may be inferred, therefore, from the manner of distribution of the magnetite, and the secondary partings existing in these rocks, that their magnetism is seated in an enormous number of small separate magnets, at least one for each of the physically distinct unit volumes.

It is a fact of observation, as will appear hereafter, that the upper surfaces of these magnetic rocks invariably attract the north end of the compass-needles (and, of course, repel the south end). From this it must be inferred that the small magnets are generally similarly oriented, and have their north ends, which would repel the north end of the compass-needle, pointing downward, and their south ends, which attract it, pointing upward. As this is the arrangement that would result from induction from the earth's magnetism, it can be concluded further (as, of course, might be assumed) that these rocks are magnetic from the earth's induction.

It is also well known, that when several bar-magnets are joined in line at opposite poles, the effect upon a compass-needle within the range of influence is nearly the same as if the joined magnets were replaced by a single magnet of the combined length. For each member of the pairs of intermediate poles, one attracting and the other repelling, is about the same distance away, and their effects so balance each other. The result, therefore, is to leave one pole, unchanged in position, and to remove the other to the end of the last magnet added. If enough magnets are added, the final result is to carry the moving pole so far away that it has no appreciable influence upon the needle. This is a condition which, from the distri-

bution of the magnetite, and the parting-surfaces which run through the magnetic rocks, must always be realized more or less completely. It is a necessary consequence of such an arrangement of the small magnets, that, in the case of a thin sheet of magnetic rock lying at a low angle of dip, the buried north-poles would not be much farther removed than the upper south-poles, and consequently the compass-needle should be relatively only slightly disturbed. This is precisely what is found to be the case.

There is thus firm ground for the conception of the magnetic rocks as made up of sheaves of small magnets, all similarly oriented in a general way, and all having their south-poles upward at or near the rock-surface, while the effective north-poles, by the continual addition of similarly oriented sheaves below, are carried down, when the rocks are vertical or nearly so, to such depths that their influence is greatly diminished or altogether imperceptible. In equal small areas the individual magnets are no doubt of very unequal number or strength. This can be proved by holding a swinging needle close to the surface of a magnetic rock, shifting its position without moving it out of the parallel plane, and observing the changes in the pointing. These are almost always large, and are undoubtedly due to the variations in strength of small areas of the upper poles. In consequence of the law of magnetism, by which the attraction (or repulsion) varies inversely as the square of the distance, the areas immediately surrounding the needle are very much more important factors in the resultant than those further removed. When the needle is held higher up, or, what is the same thing, the magnetic rock is buried, the effects are much more regular, since a larger number of the unit-areas enter into the resultant with equal weight due to equal distance, and the extremes of individual variation are lost in the general mean. Since successive magnetic cross-sections over buried rocks show on the whole a great degree of regularity, we can finally conclude that the magnetic force of these rocks is seated in an immense, practically an infinite, number of small magnets, which furnish free magnetism at the upper and lower bounding surfaces of the magnetic formation, and that on the average there is about the same number, of about the same aggregate strength, or, in other words, equal intensity

in equal areas of these surfaces, if the areas are taken large enough.

Section IV.—The Instruments and Methods of Work.

The instruments used in this work are simple and well-known. The dial-compass is an ordinary compass, carrying a 2½-inch needle, swinging inside a circle graduated to degrees, which is further supplied with a graduated hour-circle. It is therefore a portable sun-dial. The gnomon is a thread, which is attached at one end to the center of graduation of the hour-circle, near the rear sight, and at the other to a point in the forward sight, so taken that the angle made by the thread with the plane of the hour-circle is equal to the latitude of the place. When this instrument is levelled and set up in the meridian on a sunny day, the thread will cast a shadow on the hour-circle at the correct apparent solar time, from which mean time may be determined by applying the equation of time. Conversely, if it is so set up that the shadow of the thread falls on the correct apparent time, the sights of the instrument are in the true meridian. In this position the declination of the horizontal needle may be read off from the graduated circle. At work, this instrument is mounted on a light Jacob-staff, or it may be held in the hand. The Jacob-staff, although often inconvenient to carry, is preferable, as with it the readings are all taken at the same height above the ground, and the levelling is more exact and steady. In a correctly constructed instrument, with good time, the readings may be made to half a degree. Correctness in the time, however, is indispensable to good work; and this is best secured by keeping a standard watch in camp, and referring the working watches to it daily.

The dip-compass needs no description. In geological work that form known as the Norwegian, in which the needle is pivoted on a universal joint, is not so useful as the type in which the needle is rigidly confined to one plane. In taking the readings, this plane, in which the needle is free to swing, is made to coincide with the vertical plane determined by the pointing of the horizontal needle. The circle is graduated to single degrees, and with skilful work the readings are reliable to one or two degrees. It may be added that the south end is weighted

in order, either partly or completely, to balance the vertical component of the earth's force. It was found better not to balance it completely, but only to such an extent that the north end of the needle would dip about 10° (the graduation-zeros being horizontal) in an area removed from local disturbances. It is no doubt desirable that all the dip-needles used in the same work should be brought to approximately the same index-error, in order that the readings may be more directly comparable. In practice, however, it was found quite impossible to keep our three needles in unison, on account of the rough usage to which they were unavoidably subjected. As, however, the form of the dip-curves is really the object sought, and since these, in the presence of considerable disturbances, are sensibly independent of small differences in the index-error, it is not indispensable that the needles should be exactly together.

These instruments are simple, and, of course, do not give precise results. But the observations are rapidly and cheaply made, and to a sufficient degree of accuracy for the end in view. It must be stated again that the object is to detect and compare relative magnetic disturbances, and to find out the bearing of these disturbances on the distribution and attitude of the rocks which produce them. For this purpose the instruments are exceedingly well adapted.

The field-work was carried out by parties of two men each, one of whom, a skilled woodsman, carried along the line and observed the horizontal needle, while the other read the dip-needle, kept the notes and attended to the geology. According to the general plan of the field-work, a series of parallel lines was run either north and south, or east and west, across each section. The direction of the lines of travel was chosen so as to cut the strike of the rocks at the largest angle. The probable direction of strike for each day's work could be inferred in advance from what had gone before. If it were more nearly north-and-south than east-and-west, the traverse lines were run east-and-west, and *vice versa*. These directions were in many cases not the most desirable for the magnetic work alone, but the choice was practically limited by the lines of the United States Land Survey, which give for each square mile eight points of departure (at the four corners and four quarter-posts of each section), which are generally identifiable on the

ground. On these lines of travel the instruments were read at various intervals, from 5 or 10 to 100 paces, depending upon the local complications. The intervals between the lines varied from $\frac{1}{16}$ to $\frac{1}{4}$ of a mile, and were determined not only by the magnetic complications, but by the character of the surface; it being especially desirable that the ground should be so closely covered that no outcrop could escape detection. The distances along and off the lines of travel were measured by pacing. The general accuracy of the pacing is remarkable, and is essentially within the platting error of the scale of the maps. The average closing error for August, 1892, during which about 100 miles of traverse-lines were run, was 20 paces per mile, or 1 per cent. Two-thirds of the errors averaged 10 paces per mile, or 1 in 200, while the maximum was 1 in 30. But this was better than the average for the season.

The observations at each station consisted in a reading of the horizontal and dip-needles. When there were no local magnetic disturbances in the neighborhood the horizontal needle would come to rest in the magnetic meridian, which in this region is about N. 2°–3° E., or almost coinciding with the true meridian. The dip-needle, when held in thes ame meridian, would indicate the index-error. When, however, disturbing material was present the horizontal needle would point to the east or west of the magnetic meridian, at an angle determined by the direction of the resultant of the horizontal components of the earth's and the local forces. The dip-needle would come to rest in the same vertical plane, at an angle with the horizon determined by the amount and direction of the three forces, the whole pull of the earth's force, the whole pull of the local forces, and the balancing weight, and in general would show a downward deflection. After making and recording the set of observations at a station the party proceeded to the next, and so on to the end of the day. At the end of each day, or as soon as possible afterwards, the day's work was platted on a large-scale map, on which the readings of the horizontal needle were represented by short arrows drawn through the stations, turned east or west of the true meridian as the case might be, and carrying the amount of declination written at the arrow-point. The dip-observations were laid off to scale immediately below the stations, measuring all from the same

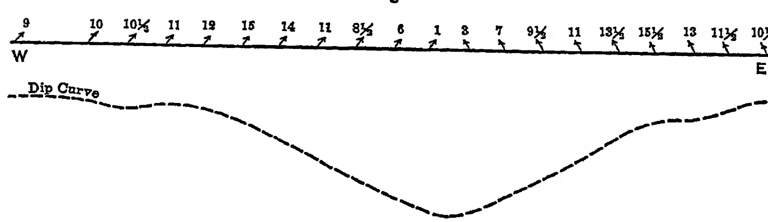
horizontal line, and the points thus established were connected by a free-hand curve.

Section V.—Facts of Observation and General Principles.

1. *Observed Deflections when the Strike is North and South and the Dip Vertical.*

If a magnetic rock, striking north-and-south and dipping vertically, is crossed by an east-and-west traverse, it is found, as the disturbing belt is approached, say from the western side, that the horizontal needle points towards the east of north, and that this easterly pointing gradually increases to a maximum.

Fig. 1



Continuing east from the maximum point the easterly declination decreases, and soon a station is reached at which the needle points due north. Still farther east the declination changes to westerly, and soon thereafter reaches a westerly maximum, beyond which again the westerly pointing in its turn gradually decreases, until finally the needle reaches its normal easterly declination, after passing through a second zero. The dip-needle readings at the same stations generally increase slowly at first, and then rapidly, and soon reach a maximum at the first zero-point between the converging arrows; beyond this to the east they decrease correspondingly, so that the dip-curve is symmetrical east and west of the maximum. These statements will be made clear by a reference to Fig. 1, which represents an actual traverse in T. 45 N., R. 31 W.

2. *Deflections of the Horizontal Needle.*

Let us now study these results more closely and see the reasons for this peculiar behavior. It has been shown that there is good reason to believe that the magnetic rocks are made

up of sheaves of small magnets, the south poles of which are all situated at or near the rock-surface, while the north poles, when the rocks stand at high angles, are carried far below the surface, so that they exert no influence upon the compass-needles; and furthermore it has been assumed that equal superficial areas of these rocks contain on the whole about equal numbers of these small magnets.

It is evident that, in crossing a rock-belt which stretches away indefinitely in both directions, only a limited part of it

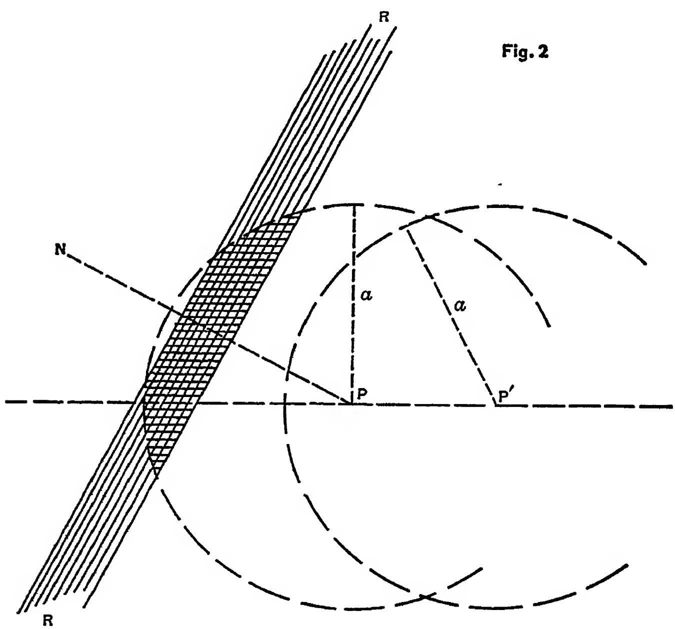


Fig. 2

will affect the readings on a given cross-section. Since the pull of the poles of a magnet on a compass-needle diminishes with the square of the distance of separation, it follows that the limits to the material that would noticeably disturb comparatively insensitive instruments would soon be reached. If we consider for the moment only the horizontal components, and call (Fig. 2) the distance a at which the needle would respond to the attraction of material, possessing the magnetic force of that with which we are dealing, then at any station, P, the material inside a circle drawn with P as a center and radius a (shaded in the figure) would exert force on P, the material out-

side would not. If the circle drawn from a station, P′, does not reach to the magnetic belt, the needles at P′ will not be disturbed.*

For reasons of symmetry it is seen that the attraction of the magnetic belt would act along the line P N, drawn through P perpendicular to the strike of the rock. Since there is as much material on one side of this line as there is on the other, the components perpendicular to it will balance each other, and the instruments at P will be affected exactly as if all the attracting material were concentrated along this line. The horizontal needle will take a position in the line of the resultant of the two forces which act upon it—namely, the horizontal component of the earth's magnetism (which acts in the line of the magnetic meridian) and the horizontal component of the material within the circle of attraction (which acts along the line P N). The force which deflects the needle from its general local direction is the component along P N, and it is evident that the greater this component the greater will be the deflection of the needle, since the direction in which it acts always remains the same at all stations for any given direction of strike of the rock. For if β is the strike of the rock measured from the north, and H and H′ are the horizontal components of the earth's and the rock-force respectively, it is readily shown that δ, the angle of deflection of the horizontal needle at any station, P, is given by the equation:

$$tan\ \delta = \frac{cos\ \beta}{\frac{H}{H'} \pm sin\ \beta} \qquad (1.)$$

From this equation it is easily seen that, no matter how great may be the horizontal component of the force of the magnetic rock, the horizontal needle cannot be deflected past the normal.

As P moves towards the magnetic belt the horizontal com-

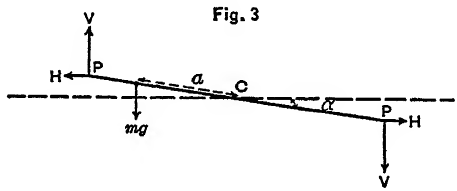
* The actual distances at which disturbances of the needles can be detected are exceedingly variable, since they depend (as will be shown hereafter) not only upon the lithological character of the magnetic formation, but also upon its strike, dip, thickness, extent and nearness to the surface. One formation in which all the conditions are exceptionally favorable distinctly deflects the dial-compass needle at a distance of 3½ miles.

ponent at first increases, and with it the westward deflection of the needle. Finally, the maximum westerly deflection is reached, beyond which the needle begins to return; it is evident, therefore, that at this point the horizontal component has reached a maximum value.

3. *Deflections of the Dip-Needle.*

The balanced dip-needle (*i.e.*, without index-error), in an area of no local disturbance, is in equilibrium under the action of two couples, viz.: the vertical component of the earth's magnetism, and the added weight. When displaced from the position of equilibrium, the horizontal couple restores it.

Fig. 3



In Fig. 3 let PP be a balanced dip-needle which has been displaced through the angle α. At the two poles the attraction and repulsion of the earth's magnetism may be resolved into horizontal and vertical components, H and V.

Taking moments about C, we have, if $\alpha = 0$, the needle in equilibrium under the couples,

$$V \,.\, 2b - mg \,.\, a = 0,$$

where $2b = PP$, $mg =$ the added weight, and a its distance from the center.

If depressed through the angle α the left-hand couple $H.\ 2b.\ \sin \alpha$ comes into play, and tends to carry the north end of the needle upward. When $\alpha = 0$, this couple reduces to 0. When α changes sign, or the north end of the needle passes above the horizontal line, $H \,.\, 2b.\ \sin \alpha$ becomes a right-hand couple, and acts to depress the north end of the needle. So the needle will finally come to rest through friction in the horizontal line, where $\alpha = 0$.

If this needle, so balanced, is carried to a station within the

influence of a magnetic rock, its dip will be determined by the composition of the new forces with the old. The vertical plane will be that in which the horizontal needle points at the same station. The equations above give us a ready means of determining the angle of dip in terms of all the forces.

Suppose the needle finally comes to rest at the angle α with the horizontal (the north pole being depressed). Then

$$V_r \,.\, 2b \,.\, \cos\alpha - mga \cos\alpha - H_r \,.\, 2b \,.\, \sin\alpha = 0, \qquad (2.)$$

where H_r and V_r signify the resultants of the horizontal and vertical components of the earth's force and the local force.

Equation (2) is easily reduced to

$$\tan\alpha = \frac{2b \,.\, V_r - mga}{2b \,.\, H_r}, \qquad (3.)$$

If the dip-needle is not balanced, but has, where there is no local disturbance, a constant index-error θ (measured from the horizontal), it is readily seen that

$$\tan\theta = \frac{2b \,.\, V - mga}{2b \,.\, H}, \qquad (4.)$$

where H and V are the horizontal and vertical components respectively of the earth's force. In an area of local disturbance the angle of dip, α, is given by equation (3). Since V and V′ always act in the same line,

$$V_r = V \pm V'.$$

Substituting this and the value of mga from equation (4), equation (3) becomes

$$\tan\alpha = \frac{\pm V' + H \tan\theta}{H_r} \qquad (5.)$$

If the index-error is θ', and the corresponding deflection α' we have

$$\frac{\tan\alpha}{\tan\alpha'} = \frac{\pm V' + H \tan\theta}{\pm V' + H \tan\theta'}$$

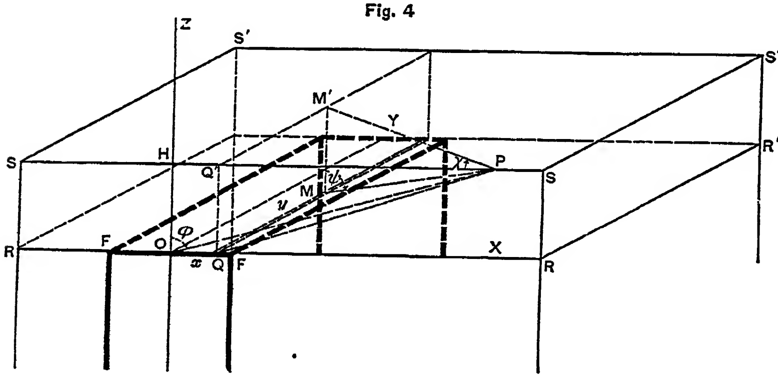
Therefore at the same station, the greater the index-error, the greater is the angle of dip in the same or two similar instruments. It is also evident that the greater the vertical component of the pull of the rock, the less will be the difference between the deflections in the two cases.

From an inspection of equation (5) it is seen that $\tan \alpha = \infty$, or $\alpha = 90°$, only when $H_r = 0$. H_r, which is the resultant of the two horizontal components, H and H′, of the earth's and the rock-force, is in general given by the equation

$$H_r = \sqrt{H^2 + H'^2 \pm 2\, H\, H' \sin \beta},$$

where β is the strike of the rock measured from the north. H_r can therefore equal zero only when $\beta = \frac{\pi}{2}$, or the rock strikes east and west, and at the same time H′ is numerically equal to H,

Fig. 4



and acting in the opposite direction. Dips of 90° cannot occur in other cases, no matter how strong the magnetic force of the rock may be. It is also evident that in general H_r has its minimum value when $H' = -H \sin \beta$. When the rock strikes north and south, or $\beta = 0$, H_r is a minimum when $H' = 0$.

4. *Horizontal and Vertical Components When the Magnetic Rock Dips Vertically.*

Let us now determine the horizontal and vertical components of the rock-force, assuming that the magnetic rock has a uniform strike in any direction, a vertical dip and a surface-width or thickness equal to $2a$.

In Fig. 4 S . . . S′ is the plane of the surface of the ground, and R . . . R′ the plane of the rock-surface. F . . . F is the magnetic formation, which is cut by the vertical plane S S R R, passed normal to the strike of the rock through any station of observation P. O is the middle point of the magnetic formation at the intersection of this plane with the rock-surface, and is the origin of co-ordinates. O X and O Y lie in the rock-surface, respectively perpendicular and parallel to the strike, and O Z is measured vertically upwards. $OH = h$ is the depth of surface-covering over the rock-surface, and is assumed to be uniform. Since the magnetic formation dips vertically, its magnetic force is seated in its upper surface, and attracts the north-ends of the compass-needles. If M is any point of this surface, the co-ordinates of which are x and y, then, if ω is a constant, the pull of the elementary area at M on the north end of a compass-needle at P is,

$$d\,M = \omega \frac{dx\,dy}{\rho^2},$$

where $\rho = MP$. If $M'MP = \psi$, the horizontal and vertical components of this pull in the plane M′MP are

$$d\,H = \frac{\omega\,dx\,dy\,\sin\psi}{\rho^2}$$

$$d\,V' = \frac{\omega\,dx\,dy\,\cos\psi}{\rho^2}$$

The resolved part of dH perpendicular to the strike is

$$d\,H' = \frac{\omega\,dx\,.\,dy\,.\,\sin\psi\,\cos\chi.}{\rho^2}$$

If $Q'QP = \varphi'$, $QP = \rho'$, $HOP = \varphi$, and $OP = r$, we have,

$$\sin\psi = \frac{(\rho^2 - h^2)^{\frac{1}{2}}}{\rho}; \quad \cos\psi = \frac{h}{\rho}; \quad \cos\chi = \frac{\rho'\,\sin\varphi'}{(\rho^2 - h^2)^{\frac{1}{2}}};$$

$$\rho'\,\sin\varphi' = h\,\tan\varphi - x.$$

$$\therefore \frac{H'}{\omega} = \iint \frac{dx\,dy\,(h\,\tan\varphi - x)}{[h^2 + (h\,\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$

$$\frac{V'}{\omega} = h \iint \frac{dx\, dy}{[h^2 + (h \tan \varphi - x)^2 + y^2]^{\frac{3}{2}}}.$$

Integrating first with respect to y between $+\infty$ and $-\infty$, and then with respect to x between $+a$ and $-a$, we have

$$\frac{H'}{\omega} = \log \frac{h^2 + (h \tan \varphi + a)^2}{h^2 + (h \tan \varphi - a)^2}$$

$$\frac{V'}{\omega} = 2 \left\{ \tan^{-1} \frac{h \tan \varphi + a}{h} - \tan^{-1} \frac{h \tan \varphi - a}{h} \right\}$$

If $x = h \tan \varphi$, and now refers to P, these equations may be written

$$\frac{H'}{\omega} = \log \frac{h^2 + (x + a)^2}{h^2 + (x - a)^2} \quad . \quad . \quad . \quad . \quad . \quad (6.)$$

$$\frac{V'}{\omega} = 2 \left\{ \tan^{-1} \frac{x + a}{h} - \tan^{-1} \frac{x - a}{h} \right\} \quad . \quad . \quad (7.)$$

In equation (6) $\frac{H'}{\omega} = 0$ when $x = 0$; and it therefore follows that a point of no deflection of the horizontal needle is found vertically over the middle point of the magnetic rock. It is also evident that at corresponding stations on opposite sides of the middle point, the horizontal components are equal, but act in opposite directions.

To obtain the points of maximum or minimum values of the horizontal component, we differentiate the right-hand side of equation (6) with respect to x, and place the result equal to zero. This gives

$$x = \pm \sqrt{h^2 + a^2} \quad . \quad . \quad . \quad (8.)$$

which determines the two points, at equal distances from O on opposite sides of the rock, at which the horizontal component has maximum values. Writing for x the measurable distance d, and squaring, we have

$$d^2 = h^2 + a^2 \quad . \quad . \quad . \quad (9.)$$

The thickness of the magnetic formation is therefore always less than the distance between the points of maximum horizontal deflection, except when $h = 0$, or the rock is uncovered,

in which case the thickness and the separation of the maxima are the same.

By differentiating the right-hand side of (7), with respect to x, it is easy to show that V' has a maximum value when $x = 0$. When the rock strikes north and south this also corresponds to a minimum value of H_r, as has already been shown; and, therefore, by a reference to equation (5) it is readily seen that a point of maximum dip coinciding with a point of no horizontal deflection is in that case found over the middle plane of the buried magnetic rock.

Where the strike is inclined to the meridian, the points of maximum dip and zero deflection will not coincide, since the maximum value of V' does not occur at the same station as the minimum value of H_r. As has been already shown, H_r is a minimum when $H' = H \sin \beta$ (β being the angle of the strike), and this is in general on the side of the rock on which the angle made with an east and west traverse is obtuse. The point of maximum dip will be situated on the same side of the rock between this station and the point of no horizontal deflection, and will approach the latter as the strike approaches the meridian, and also as V' increases relatively to H'. With strongly magnetic rocks the points of no deflection and maximum dip practically coincide on maps platted to the scale of 4 inches to the mile, except where the strike is nearly east and west.

5. *Horizontal and Vertical Components When the Magnetic Rock Dips at any Angle.*

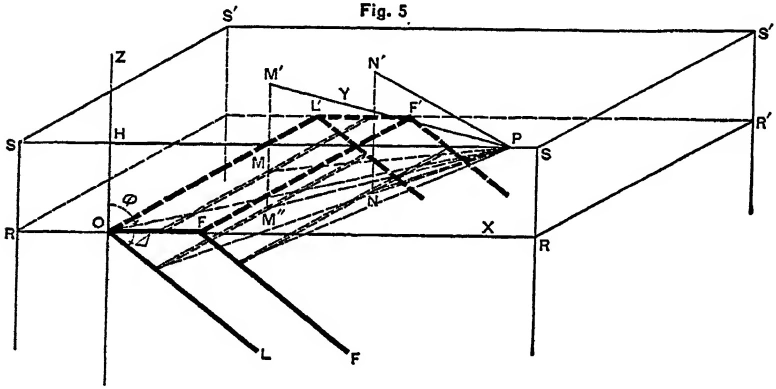
In the last article it was assumed that the magnetic rock was dipping vertically, and that it continued indefinitely downward at this angle. In consequence of this assumption, and also of the conception of the manner in which magnetism is distributed through the magnetic rocks, it has been concluded that the north poles of the rock, which repel the north end of the compass-needle, are situated so far below the surface that their effect may be neglected. Therefore we have taken into account only the south poles, which are situated at the rock-surface.

In the case of rocks which do not dip at high angles this assumption cannot safely be made, and the influence of the bottom poles must be taken into account. Since the force of

these poles acts in opposite directions from that of the upper poles, and since they are more deeply buried, it would seem that their influence in general must be to diminish the total force which acts upon the needles at any station, and therefore that the deflections both of the horizontal- and dip-needles, caused by the same rock, should be less in amount, *ceteris paribus*, where that rock dips at a low angle, than where it dips at a high angle.

In the course of the field-work certain peculiar deflections of the needles were encountered in traverses across rocks dipping at moderate or low angles. These were not thoroughly understood at the time; but the cause was believed to be connected

Fig. 5



with the angle of dip of the rock. For example, it was found along traverses crossing certain north-and-south striking rocks, which were known to have a westerly dip that may have been either high or low, that the two points of maximum deflection of the horizontal needle were not situated at equal distances from the point of no deflection between them, but that the distance of the western maximum was much the shorter. It happens that in this region no east-dipping rocks occur which are so far removed from other magnetic formations as to be out of range of their possible influence, but so far as they go, traverses across these showed that the nearer maximum was situated on the eastern side of the point of no deflection. It therefore seemed probable that the cause of the inequality in the distances from the zero-point to the maxima was the dip of the rock, and that the dip was in the direction of the nearer maximum.

In Fig. 5 S . . . S′ is the surface of the ground, and R . . . R′ the rock-surface, which is buried to the uniform depth $OH = h$. O L F F′ is the magnetic formation, dipping to the right at the angle $\varDelta$, and having a surface-width across the strike of $OF = b$. $Tan\ \varDelta = \lambda$. S S R R is a vertical plane through any station of observation, P, and normal to the strike of the magnetic rock. O, at the intersection of this plane, the lower surface of the rock, and the rock-surface, is the origin of co-ordinates, which are positive in the directions given in the figure. It is assumed that the upper surfaces of the rock, O F F′ L′ and F F F′, attract the north end of a compass-needle held at P, while the lower surface, O L L′, repels it.

Proceeding as in the last article, it is easy to show that the components of the rock-force which deflect the compass-needles at P, are

$$\frac{H'}{\omega} = \iint \frac{dx\,dy\,(h\tan\varphi - x)}{[h^2 + (h\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$
$$+ \iint \frac{dx\,dy\,(h\tan\varphi - x)}{[(h + \lambda x - \lambda b)^2 + (h\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$
$$- \iint \frac{dx\,dy\,(h\tan\varphi - x)}{[(h + \lambda x)^2 + (h\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$
$$\frac{V'}{\omega} = \iint \frac{h\,dx\,dy}{[h^2 + (h\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$
$$+ \iint \frac{dx\,dy\,(h + \lambda x - \lambda b)}{[(h + \lambda x - \lambda b)^2 + (h\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$
$$- \iint \frac{dx\,dy\,(h + \lambda x)}{[(h + \lambda x)^2 + (h\tan\varphi - x)^2 + y^2]^{\frac{3}{2}}}$$

Integrating first with respect to y between $+\infty$ and $-\infty$, and then with respect to x between b and 0 for the first terms in each equation; between ∞ and b for the second terms; and between ∞ and 0 for the third terms, and finally writing x for $h\tan\varphi$, we get:

$$\frac{H'}{\omega} = \frac{\lambda}{1 + \lambda^2} \log \frac{h^2 + x^2}{h^2 + (x - b)^2} + \frac{2\lambda}{1 + \lambda^2}$$
$$\left\{ \tan^{-1} \frac{x - b - \lambda h}{\lambda x - \lambda b + h} - \tan^{-1} \frac{x - \lambda h}{\lambda x + h} \right\} \quad . \quad (10.)$$

$$\frac{V'}{\omega} = 2\left\{ tan^{-1}\frac{x}{h} - tan^{-1}\frac{x-b}{h} \right\}$$

$$+ \frac{\lambda}{1+\lambda^2} log \frac{h^2+x^2}{h^2+(x-b)^2} + \frac{2}{1+\lambda^2}$$

$$\left\{ tan^{-1}\frac{h(1+\lambda^2)-\lambda(1+x)}{\lambda^2(1+x)} - tan^{-1}\frac{b+\lambda h-x}{h-\lambda b+\lambda x} \right\} \quad (11.)$$

If $\lambda = \infty$, and the co-ordinates are referred to axes in the middle of the rock, these equations reduce to equations (6) and (7).

By differentiating the right-hand side of equation (10), placing the result equal to zero, and solving for x, the positions of the stations at which H' is a maximum may be determined. This gives:

$$x = \frac{\lambda b + 2h}{2\lambda} \pm \frac{(\lambda^2 b^2 + 4\lambda^2 h^2 + 4h^2)^{\frac{1}{2}}}{2\lambda} \quad . \quad . \quad (12.)$$

Calling the difference of the roots, or the measured distance between the maxima, $2d$, and substituting for b its value $\frac{2a}{sin\ \Delta}$, $2a$ being the true thickness of the rock, we have,

$$d^2 = \frac{h^2+a^2}{sin^2\ \Delta} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (13.)$$

For rocks of high dip the distance between the maximum points is, therefore, but little greater than it would be were the dip vertical; and it increases inversely as the angle of dip.

A general algebraic determination of the points at which H' is 0, and V' is a maximum is impossible, since it involves the solution of equations of a degree higher than the fifth. By assuming numerical values for λ, h, and a (or b), however, curves expressing the relations between $\frac{H'}{\omega}$ and $\frac{V'}{\omega}$ and x can be plotted, from which the maximum and zero-points can be determined in any desired number of special cases.

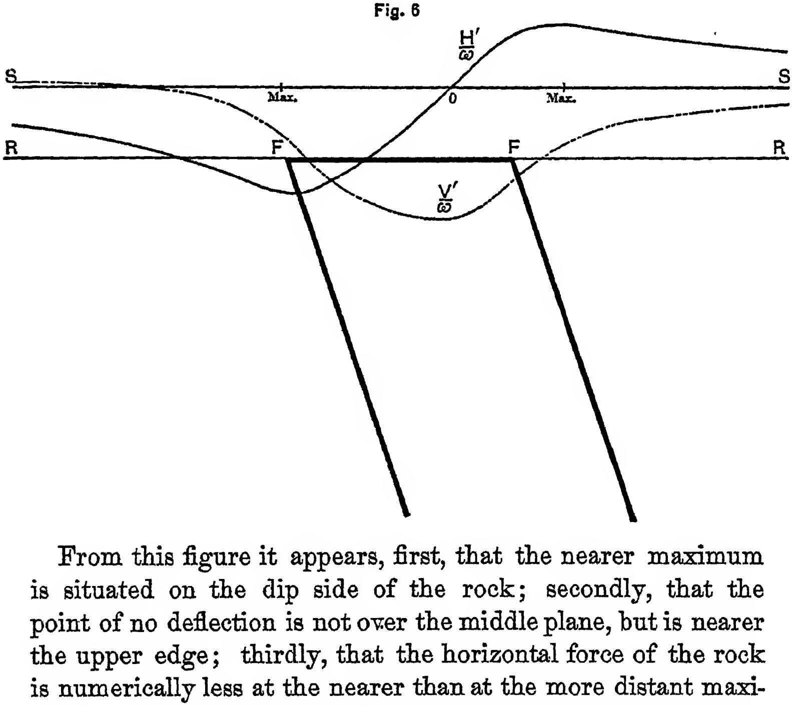
Let us first assume that $\lambda = 3$ (or that the rock dips at an angle of about 71° 34′), $h = 2$, and $a = 6$. The ordinates to the curves of Fig. 6 give the values of $\frac{H'}{\omega}$ and $\frac{V'}{\omega}$ correspond-

ing to different values of x. The ordinates to $\frac{H'}{\omega}$ do not represent the deflections δ of the horizontal-needle from the meridian, but quantities that are connected with those deflections by equation (1),

$$tan\ \delta = \frac{cos\ \beta}{\frac{H}{H'} \pm sin\ \beta}.$$

The deflections therefore vary as H' varies, and will have maximum and minimum values at the same points.

Fig. 6



From this figure it appears, first, that the nearer maximum is situated on the dip side of the rock; secondly, that the point of no deflection is not over the middle plane, but is nearer the upper edge; thirdly, that the horizontal force of the rock is numerically less at the nearer than at the more distant maximum; and fourthly, that the distance between the maximum points is nearly the same for the inclined rock as for the vertical.

In Fig. 7 the constants have the same numerical values as before, except h, which now $= 4$ instead of 2. The rock is thus buried to twice the depth of the former case. The same conclusions are true for this case as for the first. The zero-

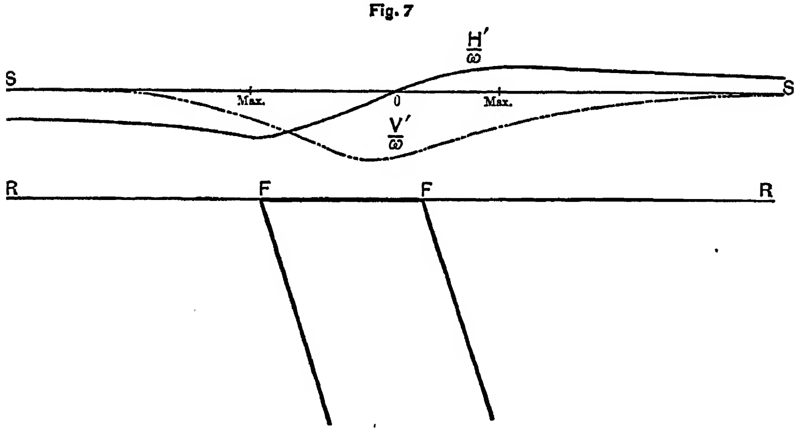
point is still nearer the upper edge of the rock, and the maxima are farther apart.

Let it next be assumed that $\lambda = 0.5$ (or that the rock dips at an angle of about 26° 34′), $h = 2$, and $a = 6$. These data lead to the curves of Fig. 8, in which, as in the case of the rock of higher dip, the maximum points are unsymmetrical to the point of no deflections, the nearer lying on the dip-side.

In Fig. 9 we have a rock of the same thickness and dip at a depth $h = 4$; and the same conclusions hold.

From these four curves, which represent formations dipping at high and moderately low angles, and buried to depths which are in the one case small and in the other great, relatively to the thickness, it is probably safe to draw the following general conclusions:

Fig. 7



1. The direction of dip of a magnetic formation is towards the nearer, and (for north-and-south striking rocks) the numerically smaller maximum.

2. The point of no deflection between the converging maxima is not situated over the middle plane of the formation, but is nearer the upper edge. But with increasing depth and diminishing angles of dip, this point may pass beyond the upper edge.

3. In the case of the rock of higher dip, Figs. 6 and 7, the deflections of the horizontal-needle on the dip-side are less, and on the other side are greater, than they would be if the rock were vertical. In the case of the rock of lower dip they are

less on both sides for a moderate depth of burial, while for a

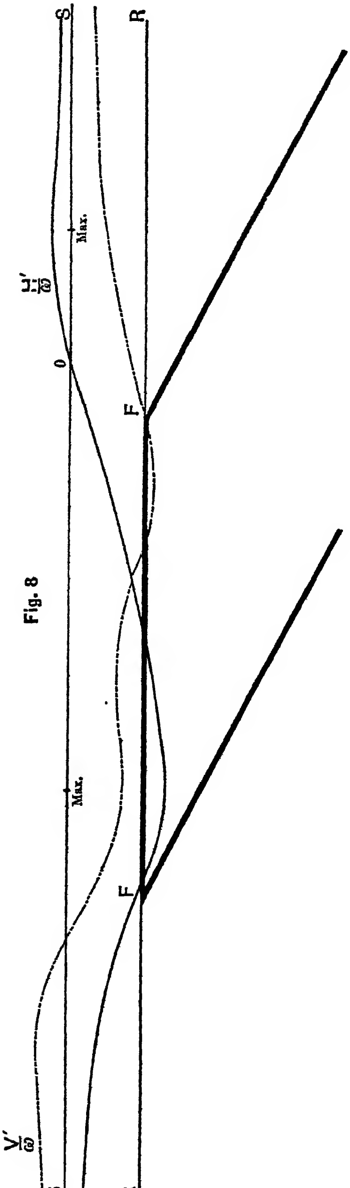


Fig. 8

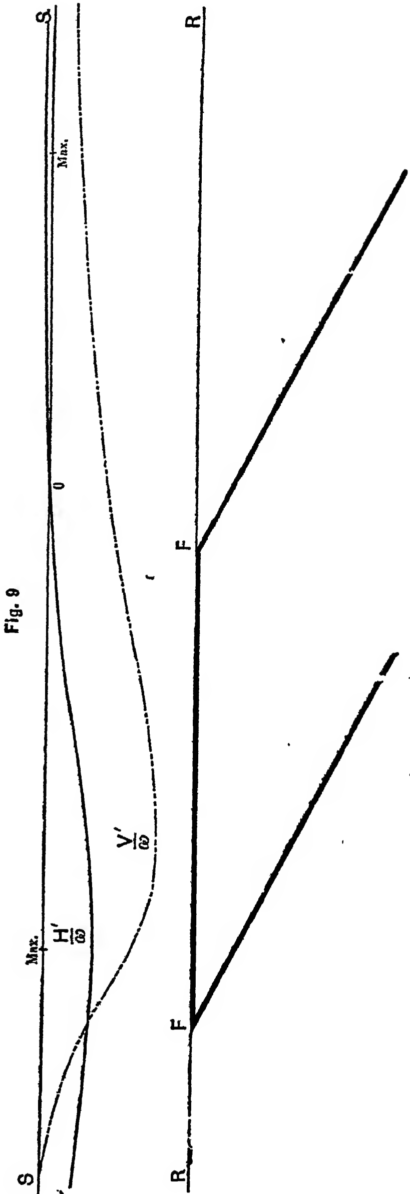


Fig. 9

greater depth they follow the same law as the rock of higher dip.

4. With slightly inclined rocks, for moderate depths of surface-covering, the disturbances are spread out over a much wider zone on each side, and the maxima are less sharp, particularly the maximum on the dip-side. Under these circumstances irregular and anomalous deflections would be expected in practice, as will be seen in the following sections.

5. The curves of the vertical component show maximum values near the zero-value of the horizontal component only in the case of the rock of high dip. In the case of the rock of lower dip, the vertical component has a negative value, or is directed upward over a wide zone on the side of the rock opposite to the dip-side. Over this zone the readings of the dip-needle will be less than normal, or even negative if $V' > H \tan \theta$. This is in accordance with the facts of observation.

6. *Determination of Depth.*

The relations between the dip and thickness of a magnetic rock, the distance between the horizontal maxima, and the depth of covering are given in equation (13). By assuming numerical values for Δ, and either for a, the half-thickness of the rock, or for d, the half-distance between the maximum points, it is easy to plot the curve which expresses the relations between the other two quantities. If d is taken as the constant, the equation represents a circle; if a, it represents a hyperbola.

This equation may have a useful application in making it possible to judge, in advance of actual test-pitting, of the probable depth of surface-covering over a magnetic rock, for which the original assumptions are fulfilled, and the numerical values of Δ, a and d are determinable. It will be remembered that the assumptions upon which equation (13) rests are the fundamental ones of Section III., and also that the rock has a uniform strike. In practice, the uniformity of the strike can be established by other traverses on each side of the one in question, and Δ, the angle of dip, may usually be determined by the outcrop of other formations in the same series.

For any practical application it is also necessary that a (half the thickness of the rock), and d (half the distance between the stations at which the horizontal component is a maximum), should be known. From an inspection of the equation it is evident that any close determination of h, except for great

depths of covering, depends upon very precise knowledge of the ratio between a and d. The practical difficulties in the way of the measurement of a, and the ever-present probability that the rock may vary from point to point, not only in actual but in effective magnetic thickness (which is what a actually signifies), make it clear that for the most part h can only be found approximately. Such variation may result from unequal distribution of magnetite along or across the strike, from inequalities of original deposition and subsequent metamorphism, or from a general swelling of the thickness due to one or many intrusions of non-magnetic igneous rock, the presence of which below a certain depth, as will be seen hereafter, cannot be detected. Also in the case of a rock striking due east and west the method fails, from the fact that $2d$ cannot be determined on the ground.

The determination of h is therefore hedged in with important limitations; yet in many cases the information supplied by the equation may be very useful. The difficulties in the way of measuring a are disposed of in the event that, along one traverse on the strike of the rock h is known, as it may be, by the sinking of a test-pit. This value of h at once gives a value of a, which may be used on other traverses across the same rock with much more accurate results, in the lack of disturbing factors, than if a were known only by measurement. It should be added, that when the traverse crosses the strike of the magnetic-rock at the angle γ, the distance d measured on the line of the traverse must be multiplied by sin γ, in order to get the value of d to be used in the determination of h, and also that h must be corrected for the height of the instrument.

More general information as to relative depths of burial is also given by the dip-curves. It is easily seen that where the superficial covering is small the vertical component of the rock-force must remain small, except immediately over the rock. This condition is, therefore, indicated by steep slopes in the dip-curves. On the other hand, where the depth of covering is considerable, the vertical component increases slowly and steadily, beginning at stations at a distance from the rock, and the resulting dip-curve approaches the maximum with gentle slopes.

7. *Summary.*—It is desirable, before proceeding to the con-

sideration of special cases, to review the results established in this section. We started with a description of the most striking phenomena exhibited by the pointings of the magnetic-needles in a particular instance, *i. e.*, when the disturbing formation had a north-and-south strike and a vertical dip. These are: (1) the convergence of the horizontal needle towards the rock for a considerable distance on each side; (2) the occurrence of two points of maximum deflection of the horizontal needle, one corresponding to each direction of convergence; (3) the occurrence of one point of no deflection of the horizontal needle, midway between these maximum points; and (4) the coincidence of the point of maximum dip with the point of no horizontal deflection. It was also shown that the points of maximum and zero deflection of the horizontal needle must correspond to maximum and zero values of the horizontal component of the rock-force.

We then took the case of a rock having a uniform strike in any direction, and a vertical dip, in which magnetic force was distributed in accordance with the fundamental assumptions of Section III., and the following deductions were reached in consequence of the general laws of magnetism: (1) that the horizontal component of the magnetic force of such a rock would have a zero value at a station in the middle plane, and two maximum values at equal distances, one on each side of this plane; (2) that the distance between these maximum points might be equal to, but in no case could be less, and is generally greater, than the thickness of the rock, and increases with an increase in the depth to which the rock is buried; (3) that the angle of dip would have a maximum value in or near the middle plane of the rock, practically at the same station at which there is no deflection of the horizontal needle; and (4) that the depth to which such a magnetic-rock is buried is a function of the thickness of the rock and the distance between the horizontal maxima, and can be determined when these are known.

We next took the general case of a magnetic rock having a uniform strike, and dipping at any angle, and showed that the fact of observation, that the two points of maximum horizontal deflection are at unequal distances from the central zero, is a necessary consequence of the fundamental assumptions.

The actual phenomena are so completely concordant with the deductions that the inferences of Section III. are strongly fortified, and the following practical conclusions may be regarded as established:

(1) The strike of a magnetic rock is given by the line joining the points on successive traverses, at which the horizontal needle is not deflected from the local meridian between the converging arrows, or at which the dip-angles are a maximum. When the rock is vertical this line lies in the middle plane of the rock and fixes its position. It may be called a line of magnetic attraction.

(2) The dip of a magnetic rock is towards the nearer horizontal maximum.

(3) The thickness of the magnetic formation must, if buried, always be less than the distance between the maximum points.

Section VI.—Applications to Special Cases.

In the preceding section certain general conclusions have been established with regard to the relative positions of the stations at which the horizontal and vertical components of the force of a magnetic rock have maximum and zero-values. The deflections produced by these components from the positions which the magnetic-needles assume under the action of the earth's force, have maximum and zero values at the same stations at which the components have maximum and zero values; and therefore the conclusions as to the relative positions of these points are true for any angle of strike. But certain numerical relations between the deflections depend upon the orientation or strike of the magnetic formation and upon the direction of dip, and these will now be considered.

1. *The Magnetic Rock Strikes East or West of North and Dips Vertically.*

Let us first take the case of a rock striking east of north. At the stations within range of the local influence on the east side of such a rock-belt, the horizontal needle is pulled west of the meridian, reaches a westerly maximum, then points north, then east of the meridian and reaches an easterly maximum. The pointings of the horizontal needle, therefore,

follow the same general course as before. It is observed, however, that the western deflections on the east side of the belt are generally not so great as the corresponding eastern deflections on the west side of the belt. The reason for this is easily seen. At each station east of the belt the local pull acts along the normal to the belt drawn through the station. This normal makes with the local magnetic meridian an acute angle. The needle will come to rest within this acute angle along the line of the resultant of the horizontal components of the two forces, the earth's and the local force, which determine its position. However strong the local pull may be, the horizontal needle cannot be deflected past the normal.

At the corresponding stations on the west side of the disturbing belt the local pull also acts along the normal from the station to the belt and has the same numerical value. But in this case the normal makes an obtuse angle with the magnetic meridian. For two points equally distant from the magnetic belt, one on the east and the other on the west, the resultant for the western point will, therefore, make a larger angle with the meridian than that for the eastern.

On the other hand, when the rock strikes west of north, it is observed that the horizontal deflections are greater on the east side than on the west, and the explanation is entirely similar to that given above.

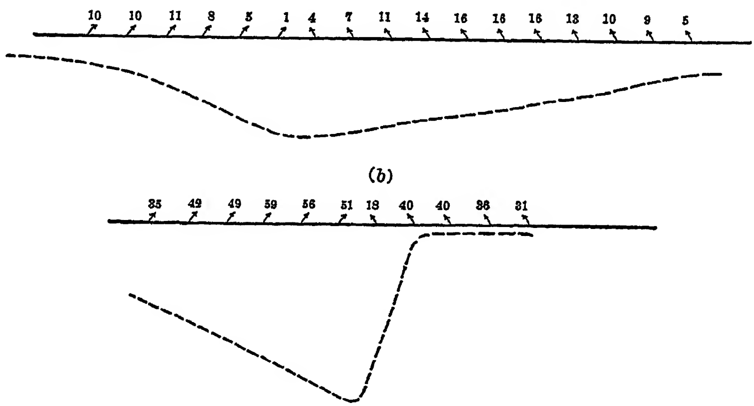
The dip-needle observations at the same stations show general phenomena quite like those in the case in which the strike of the rock coincided with the meridian. They gradually increase to a maximum near the station, where the horizontal needle stands at zero between the converging arrows, and gradually decrease from this maximum on the other side. It is noted, however, that the readings are not equal at corresponding stations on opposite sides of the maximum. When the strike is east of north the western station shows a higher dip than the eastern, while, when the strike is west of north, the eastern station shows a higher dip than the corresponding station on the west. Generally stated, then, the stations on that side of the magnetic rock, on which the angle between the strike of the rock and the line of traverse is obtuse, show greater dip-angles than the corresponding stations on the side

on which this angle is acute. As the angles of dip are represented graphically by a continuous curve, this is the same thing as saying that the dip-curve is steeper on the acute- than on the obtuse-angle side. Fig. 10 shows two such sections—(*a*) where the strike is northwest and (*b*) where it is northeast. The facts above stated are well brought out in the two dip-curves.

These phenomena are easily explained by the following considerations. The vertical components tend to lower the nee-

Fig. 10

(*a*)



dle, and would carry it to a vertical position, except for the action of the horizontal forces, which tends to keep it horizontal. At any station on the acute-angle side of the magnetic belt the resultant of the two horizontal components is larger than at the corresponding station on the obtuse-angle side (the two being represented by the longer and shorter diagonals of a parallelogram). Since the vertical forces are the same at the two stations, it follows that on the obtuse-angle side the angle of dip must be larger than on the acute-angle side. Or expressed algebraically, since H_r is the only variable on the right-hand side of equation (5),

$$tan\,\alpha = \frac{V' + H\,tan\,\theta}{H_r},$$

it is evident that tan α, and, therefore, α, the angle of dip, increases with a decrease in H_r.

The two points of maximum deflection of the horizontal needle are situated at equal distances from the middle zero, but the deflections have unequal numerical values, that which lies on the obtuse-angle side being the larger, for the reasons already given.

If the rock dips at an angle less than 90°, these results are either intensified or greatly modified, depending upon the direction of dip. It was shown in the last section that the horizontal component of the rock-force is smaller on the dip-side. If the strike and dip are both towards the same side of the meridian (*e.g.*, if the strike is northwest and the dip southwest), it is evident that the numerical difference between the deflections of the horizontal-needle on the two sides of the rock will be still greater than if the rock were vertical. On the other hand, if the strike and dip are towards opposite sides of the meridian (*e.g.*, if the strike is northeast and the dip northwest), the difference between the deflections on the two sides is less than for a vertical dip, or may even be reversed.

The deflections of the dip-needle in the case of rocks dipping at angles less than 90° are also greatly influenced by the direction of dip. If strike and dip are towards the same side of the meridian, the difference noted above in angle of dip on the two sides of the rock is neutralized, and the dip-curve tends to become symmetrical; while if they are towards opposite sides of the meridian, the difference is increased. It is to be noted that the influence on both instruments of the direction and angle of dip of the rock becomes weakened with an increase in surface-covering.

2. *The Magnetic Rock Strikes East and West.*

When a vertically-dipping magnetic rock strikes east and west, or nearly so, the traverse lines must be run north and south, so as to cross it as nearly as possible at right angles. In approaching such a belt from the south the instruments give little warning. The readings of the horizontal needle show either no deflections, or else very slight deflections, from the magnetic meridian. Past the middle of the formation the horizontal needle is strongly deflected, often through an angle of 180°, so that it may point due south. But as the magnetic rocks

having this strike which were encountered in our work were not deeply buried, and had also quite irregular upper surfaces, generally the needle pointed either east or west of south, on account of the weight which the nearness to the surface gave to the adjacent material, either from the irregular distribution of magnetite or from the protrusion of small masses above the general level. Continuing north, the horizontal deflections gradually diminish and eventually disappear.

This behavior of the horizontal needle is explained in the same way as in the preceding cases. The position of the needle at any station is determined by the resultant of the horizontal components of the two forces, the earth's force and the rock-force that act upon it. South of the magnetic rock these two components act in the same direction and essentially in the same line, since the magnetic meridian practically coincides with the true meridian. The resultant, therefore, is equal to their sum and coincides with them in direction, and consequently there is no deflection. North of the magnetic rock the two horizontal components act in opposite directions, and when they are in the same line the needle takes up its position in the direction of the greater, which determines that of the resultant; when H' is greater than H (which often happens near and north of the rock), this direction is due south. When the two components do not act in exactly the same line, the needle will point east or west of south at an angle which depends on the angle between the two forces and their ratio.

Still further north the horizontal component of the rock-force diminishes rapidly, and we consequently first pass through a zone of large and diminishing deflections to the east or west, depending on the side of the meridian on which this component falls; and finally, when it becomes insensible, the needle rests again in the meridian.

In the case of a rock striking east and west, the points at which the horizontal component of the magnetism of the rock has maximum values become indeterminate by the methods hitherto described, from the fact that throughout the traverse the two components act in, or nearly in, the same line, and the deflections from the local magnetic meridian therefore, do not

indicate the relative strengths at different stations of the horizontal component of the rock-force.

The dip-needle readings for an east-and-west striking rock are as follows: At some distance south of the rock the angles are constant at the index-error. As the rock is approached, the angles of dip depend upon the depth of burial. If the surface-covering is considerable, an increase in the dip-angles begins at a considerable distance away, and progresses continuously as the magnetic belt is approached. If the rock is near the surface, the dip-needle shows either the constant index-error or else angles of dip less than the index-error for all stations south except those very near the southern margin of the rock. The maximum reading is attained north of the middle plane of the rock, at a distance from it, which also depends upon the depth of covering. Farther north the dip-angles decrease slowly, and are in general greater than at the corresponding stations south. The form of the dip-curve, therefore, shows a steeper slope south of the magnetic rock than north of it. The reasons for these differences will be evident from the following considerations.

Let it be supposed, for the sake of simplicity, that throughout the north-and-south traverse the two horizontal components act in the same line in the meridian. At any station south of the magnetic rock they act in the same direction, and their resultant will be their numerical sum. At the corresponding station north they act in opposite directions, and their resultant will be their numerical difference. The angle of dip is given by equation (5):

$$\tan \alpha = \frac{V' + H \tan \theta}{H_r}$$

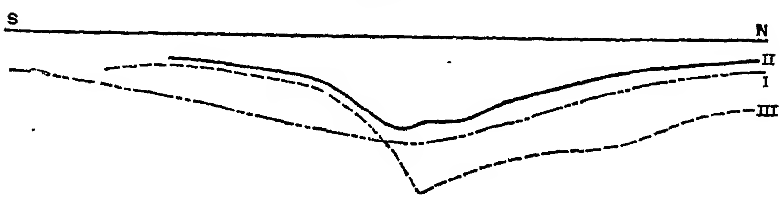
For the two corresponding stations, V' will be the same. The other quantities are all constants except H_r. For the south station, $H_r = H' + H$; for the north station, $H_r = H - H'$, where H and H' are respectively the horizontal components of the magnetism of the earth and of the rock, as before. The numerator of the right-hand side of the equation will be the same for both stations, while the numerical value of the denominator will be less for the north station than for the south.

Consequently tan a, and therefore a, will be greater for the north station.

For great depths of superficial covering, however, these differences become almost imperceptible, owing to the fact that H′ is so small that H_r is essentially the same at the two corresponding stations. The tendency therefore, as h increases, is for the dip-curve to become symmetrical. In Fig. 11, Cases II. and III. represent rocks striking east and west, buried to different depths, II. being the deeper.

In the special case in which $H' = -H$, $H = 0$, and the dip-needle stands at 90°. This can only take place north of the

Fig. 11



rock, and may, depending on the strength of H′, be found at two stations, one on either side of the station at which H′ is a maximum. At the same stations the horizontal needle is not acted on by any unbalanced force, and rests indifferently in any position.

The dips less than normal which are often observed at stations south of a magnetic rock which lies very near the surface are also easily understood by a reference to equation (7). At these stations the resultant pull of the rock is so nearly horizontal that the vertical component V′ is very small in comparison with the horizontal component H′. If V′ is a negligible quantity, equation (5) becomes

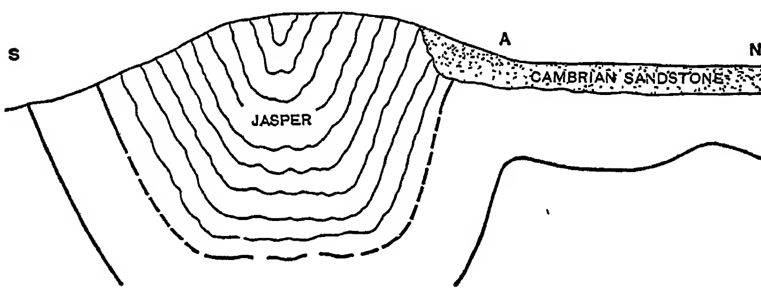
$$tan\ a = \frac{H}{H + H'} \cdot tan\ \theta.$$

In such cases, the angle of dip is therefore less than the index error. With north- or south-dipping rocks, where V′ is negative, tan a becomes negative when $V' > H \tan \theta$.

In a single locality, namely, north of the Groveland mine, we encountered the only instances met with in the whole course of our work in which the south-ends of both needles were attracted

(and the north-ends repelled) by a magnetic rock. This rock, the Michigamme jasper, occupies the interior of a shallow east-and-west-striking syncline, as has been fully proved by extensive test-pitting and diamond-drill work, as well as by abundant natural exposures. The jasper forms an east-and-west ridge, rising with steep slopes about 150 feet above the valley which borders it on the south, and about 70 to 80 feet above a flat table-land on the north. From the foot of the ridge on the north for some distance out on the table-land, the north-end of the horizontal needle pointed north with slight deflections, while the north-end of the dip-needle was actually elevated, and the south-end depressed. It was evident therefore that the south-poles of the compass-needles were attracted by the rock,

Fig. 12



and that the north-poles of the latter must be nearer the instruments than the south-poles.

These exceptional phenomena are easily understood by a reference to Fig. 12, which gives a geological cross-section and profile of this hill, with the vertical scale slightly exaggerated. The geological structure on this section has been very closely determined by many diamond-drill holes and test-pits, and the limits of the jasper are known with much precision. In consequence of the southerly dip of the northern edge of the jasper syncline, and also of the sudden drop from the ridge to the table-land, it happens that the buried north-poles along the lower boundary of the formation are brought nearer to the stations north of A than the south-poles at the actual surface of the rock. Therefore the south-poles of the needles at these stations are attracted, and not the north-poles, as is usually

the case, in consequence of an exceptional combination of structural and topographical conditions.

3. *Two Parallel Magnetic Formations.*

The cases so far considered have involved only one belt of magnetic rock, which has been assumed to have a uniform dip in one direction, or, in other words, to be a monocline. In practice, however, owing to complexities of structure and other causes, which will be considered hereafter, it frequently happens that two or more approximately parallel belts are found within the range of one another's influence. Under these circumstances the effects produced upon the magnetic needles are correspondingly complicated.

In the special case of two parallel formations, equally magnetic and buried to an equal depth, which strike north and south, and dip vertically, or at high angles, the deflections of the horizontal needle, both on the east side of the east belt and on the west side of the west belt, follow the same course as in the case of a single formation similarly situated. But in the area including the two formations and the territory between them, the deflections of the horizontal needle, in amount and direction, depend entirely upon the distance between the formations and the depth of covering over them. If the distance is considerable, and the depth to the rock-surface small, then in a traverse run from east to west, the needle, after passing a westerly maximum, will come to rest in the meridian at a point west of the middle point of the eastern rock. This point will be determined by the balancing of the two horizontal components of the local forces. The eastern rock acts at a small angle with the vertical, but with great relative energy, owing to its nearness. The western rock acts at a large angle with the vertical, but with less energy, owing to its greater distance. It is therefore evident that for certain depths, and certain distances of separation, the horizontal components of these forces will balance each other, and the needle will rest in the meridian. West of this first zero-point the needle at first will point east, since a small change in position increases the eastern angle, and consequently the eastern horizontal component, while the western angle is slightly diminished, which tends to diminish the western horizontal component at the

same time that the decrease in distance tends to increase it. The eastern deflection increases to a maximum and then diminishes as we go west, and finally becomes zero at the point midway between the two formations, when the horizontal components again balance each other. West of this middle zero the horizontal deflections are entirely symmetrical with those at the corresponding stations east, but in the opposite direction.

Therefore, under certain conditions, which are satisfied when the depth of superficial covering is small in comparison with the distance between the magnetic formations, the horizontal needle shows three points of no deflection, and four points of maximum deflection, two towards the east and two towards the west. In the territory between the outside zeros the horizontal deflections are smaller than at stations east or west of these points, at the same distance from the disturbing formations.

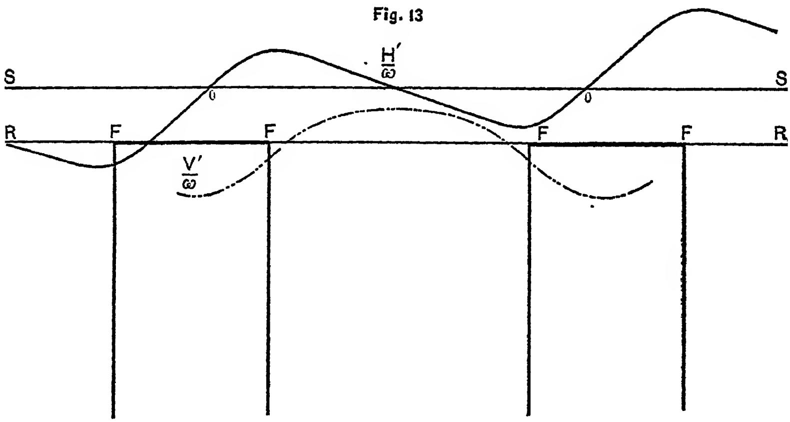
The dip-needle shows along the same traverse two points of maximum dip, which nearly correspond to the points of no horizontal deflection.

When the depth of superficial covering is large in relation to the distance between the rocks, there may be but one point of no horizontal deflection, viz.: that at the station midway between them. In this case the back pull of the eastern rock is not equal to the forward pull of the western rock, and the latter, consequently, is not balanced except at the point mentioned. Here the two inside points of maximum deflection vanish, or are represented by variations in the rate of change in the pointing. The dip-curve should show either one maximum at the center or two maxima, one on each side of the center, the distance between which depends on the ratio of the depth of burial to the distance between the rocks.

The analysis of these cases by the general method of Section V. presents no difficulties. For the purposes of illustration, however, it is sufficient to consider two extreme cases, and to represent the values of $\frac{H'}{\omega}$ for these graphically. Let it first be assumed that the distance between the rocks is 8, and that $a = 3$, and $h = 2$. This represents the conditions when the distance of separation is large compared with h. The ordinates to the curve of Fig. 13 give the values of $\frac{H'}{\omega}$ which correspond to

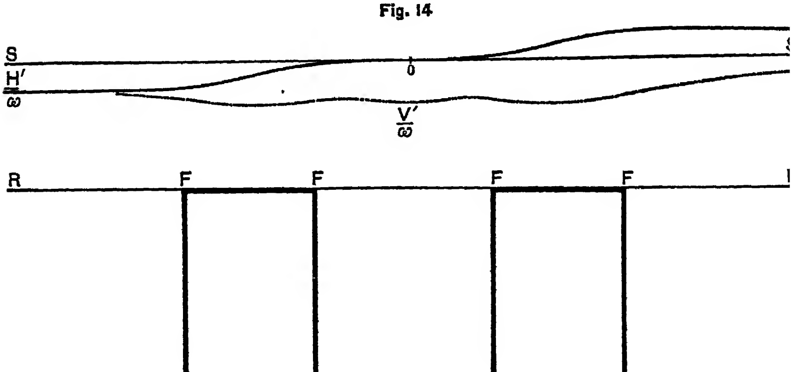
the different stations of observation. Those parts of the curve which are above the horizontal axis of co-ordinates represent the portions of the traverse in which H′ is directed towards the west; the parts below, those in which it is directed

Fig. 13



towards the east. It is seen, as already stated, that besides the middle point there are two other points of no horizontal deflection, which do not exactly correspond with the points vertically over the middle of the magnetic formations, but are somewhat

Fig. 14



nearer the adjoining edges; and also four points of maximum deflection, one on each side of each rock. The maximum points inside of the two formations have smaller deflections than those outside, as shown by the relative lengths of the ordinates to the

curve, and also between the inside maximum points the horizontal components are directed away from the middle point.

The curve of $\frac{V'}{\omega}$ represented by the dotted line, has two maximum values, which fall nearly over the two rocks.

If we next assume that the distance between the rocks is 8, and that $a = 3$, and $h = 8$, we obtain the curve of Fig. 14, which represents the value of $\frac{H'}{\omega}$ when h is large compared with the distance of separation. This case shows but one point of no horizontal deflection, between the two rocks, and but two points of maximum deflection, one on the outside of each.

The curve of $\frac{V'}{\omega}$ represented by the dotted line, shows the interesting feature of three maximum points, one at the center, and one over each rock. If h were relatively a little greater, these would evidently coincide at the center.

These curves illustrate the importance of the relation of depth to distance of separation as the controlling factor in the phenomena. If the horizontal distance could be supposed to remain unchanged and the depth to vary, we have in the corresponding traverses a series, at one end of which, for a very small depth, the observations represent essentially the phenomena due to separate formations beyond each other's influence, while at the other, for great depths, they cannot be distinguished from those caused by a single formation.

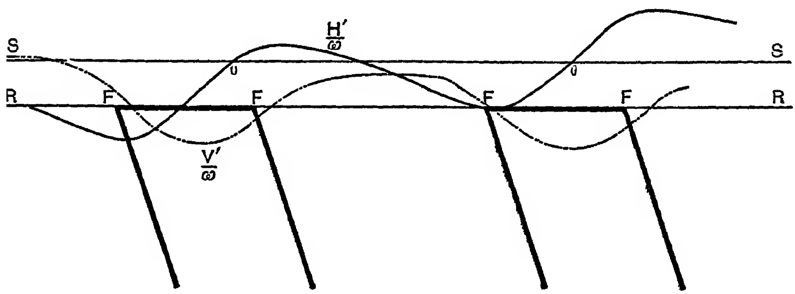
If the two parallel formations are not vertical, but dip in the same direction at the same angle, the resulting curves are somewhat different. Figs. 15 and 16 show two cases in which the elements are the same, except the depth of covering and the thickness of the intervening non-magnetic material. Here the rocks dip at an angle of 71° 34′, and the width at the rock-surface is 6.3 for each.

In Fig. 15, where $h = 2$ and the width of the non-magnetic bed is 10.7, and the covering is, therefore, relatively small, the presence of two rocks is distinctly shown by the curves of both components, and the chief result of their interaction is to introduce an additional point of no horizontal deflection between them, on each side of which the horizontal arrows diverge.

The positions of the maximum and of the other zero-points are hardly disturbed, and consequently the direction of dip is very clearly indicated.

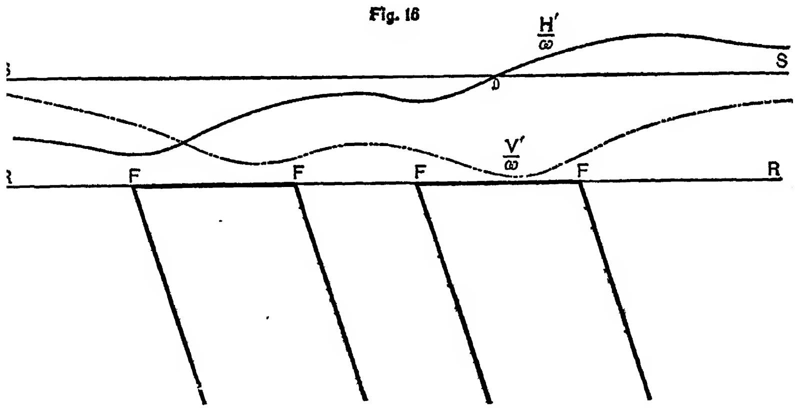
In Fig. 16, where $k = 4$ and the formations are separated by non-magnetic material 4.7 wide, there is but one zero-point,

Fig. 15



nearly over the middle of the upper formation, towards which the pointings of the horizontal-needle converge. West of this are two points of maximum eastern deflection, between which

Fig. 16

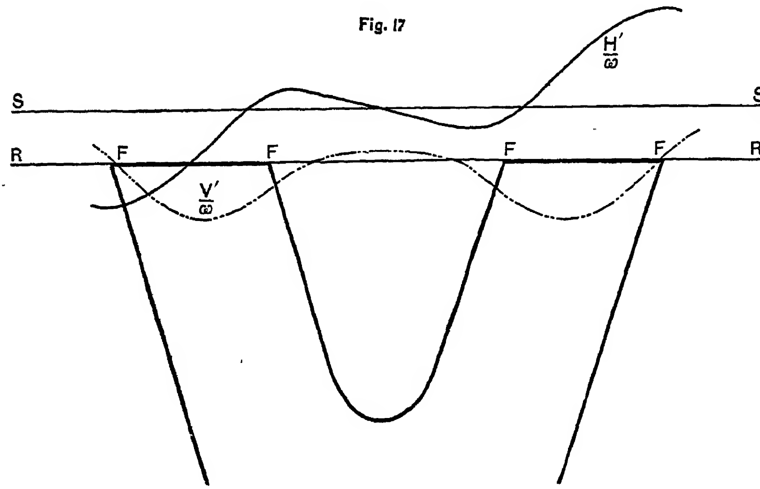


a faint minimum represents the backward pull of the lower formation.

If the two magnetic formations are parallel in strike, but dip towards each other at equal angles, the resulting curves of the two components are shown in Figs. 17 and 18. Fig. 17 illustrates the effects of a syncline with steeply dipping sides, the superficial covering being relatively shallow. These conditions result in a point of no horizontal deflection over the middle of

the trough with diverging arrows on each side, and besides a point of no horizontal deflection over each rock, towards which the arrows converge. The positions of the two maximum points for each rock, and of the zero between them, is nearly the

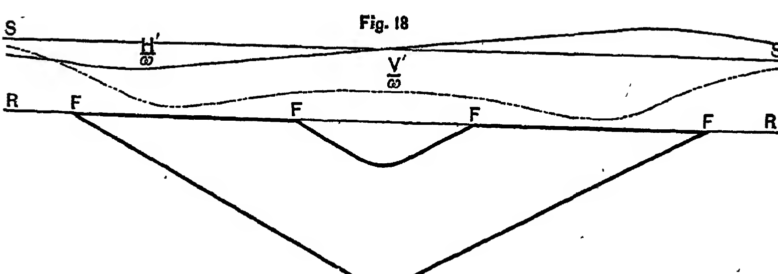
Fig. 17



same as if the other rock were absent, and consequently the fact that the rocks dip towards each other is clearly indicated by the unsymmetrical distances.

In Fig. 18 the depth of the rock-surface is much greater relatively to the inside distance between the legs of the syncline,

Fig. 18



tively to the inside distance between the legs of the syncline, and the dip is flatter. In this case there are but two points of maximum deflection, one on each side of the syncline, and but one point of no deflection, over the middle of the trough. The maximum points represent the outside maxima of the former case, and the result of the interaction of the two legs is to in-

crease the numerical values of these, as well as to bring them nearer together. It is evident that the deflections of the horizontal-needle in this case could hardly be distinguished from those that would be produced by a single vertical formation buried to a considerable depth.

Let it next be supposed that the two rocks dip towards each other at different angles, the strikes remaining parallel, and also that the rock of lower dip is buried to the greater depth. This then is a case in which the magnetic effect of one limb of the syncline is much stronger than that of the other.

It is evident that with different relations of depth and separation we may have very different series of deflections of the needle. With slight depth and large relative separation, there would be three points of no horizontal deflection, the middle point being unsymmetrical to the other two, and lying nearer the rock of weaker attraction. Of the two outside points, that which represented the stronger rock would be indicated by the more pronounced convergence of the horizontal deflections on both sides. The dip-needle would show two points of maximum dip, nearly coinciding with the two points of no horizontal deflection. These would be of unequal value, that over the stronger rock being the greater.

On the other hand, with the depth large in relation to the distance between the magnetic belts, but one point of no horizontal deflection would result, which would be situated near the stronger rock. The corresponding point over the weaker rock would be indicated by a retardation or reversal in the rate at which the horizontal deflections approached the maximum due to the stronger rock, on the same side. For example, if the weaker rock were west of the stronger, a cross-traverse from west to east would show west of the western rock a constantly increasing easterly deflection of the needle. After passing the western rock, the horizontal needle would still point east, but at a smaller angle, owing to the westerly pull of the horizontal component of the western rock. Farther east, the easterly pointing would again increase up to the maximum easterly pointing due to the stronger eastern rock, at a station west of the point of no deflection. In Fig. 19 the curves of the two components are given for the special case in which the right-hand limb of the synclinal dips at an angle of 90°, and has a

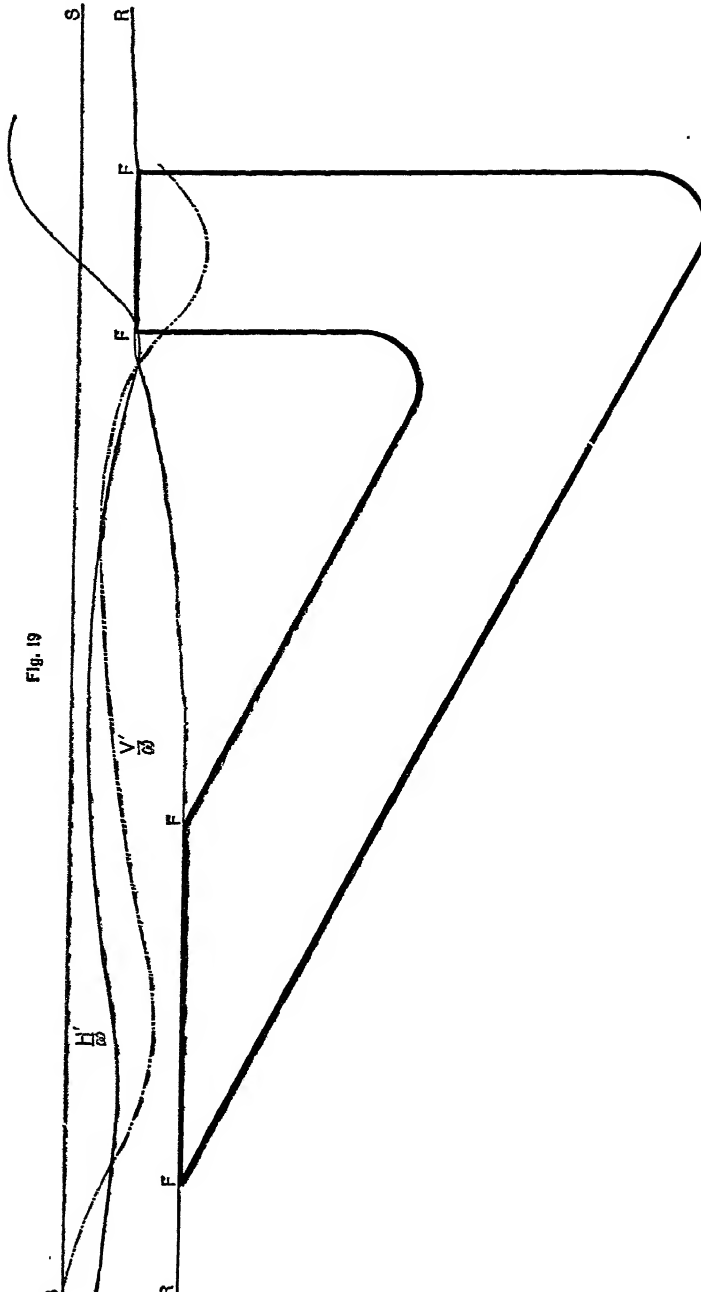
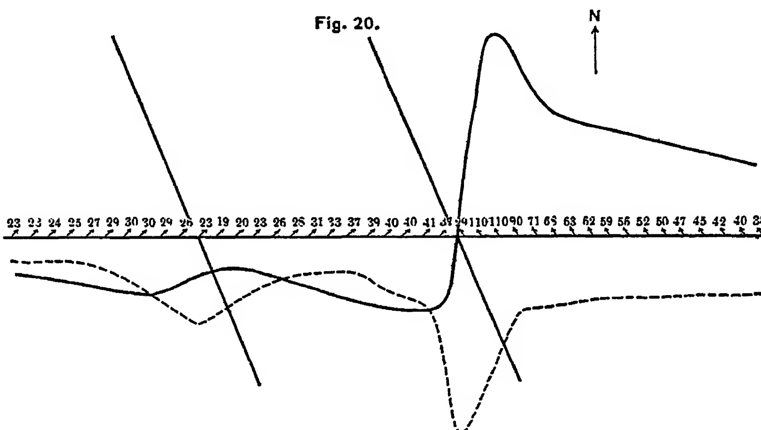


Fig. 19

surface-covering $h = 2$, while the left-hand limb dips at an angle of $26° 34'$, and has a surface-covering $h = 4$. It is interesting to compare the theoretical results of this figure with the curves of Fig. 20, which represent deflections actually observed, and not components.

In the latter figure the strike of the two rocks is represented by the heavy lines. The two rocks are the same formation, brought up by folding on opposite sides of a synclinal trough. The synclinal is slightly pushed over, so that the eastern rock dips nearly vertical, while the western has a much lower dip toward the east, and is also more deeply buried. These facts

Fig. 20.



rest on independent evidence, yet they might all be inferred from the observations recorded in this figure.

The dip-curve in this case shows two distinct maxima, a smaller under the zone of retardation, and a larger over the point of no horizontal deflection, which correspond respectively to the two magnetic rocks.

If the two formations are parallel in strike, but dip away from each other, the curves of the horizontal and vertical components for different angles of dip and different relations of thickness and depth of covering are shown in Figs. 21 and 22. In Fig. 21 the formations are widely separated, h is relatively small, and the angles of dip are equal and low; the interaction of the two rocks therefore extends over a narrow zone only, and the curves of the components clearly indicate the presence of two formations and the direction of dip of each. In Fig. 22

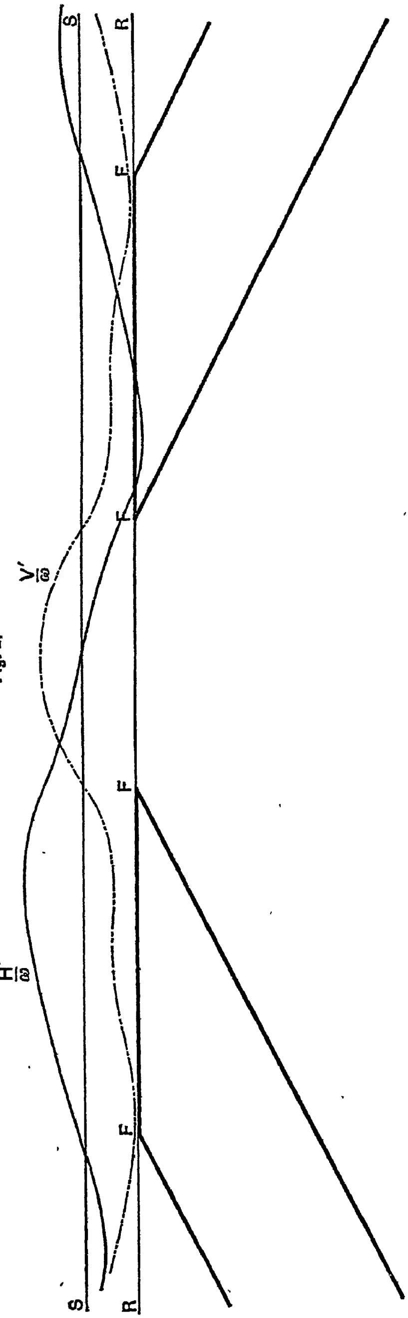
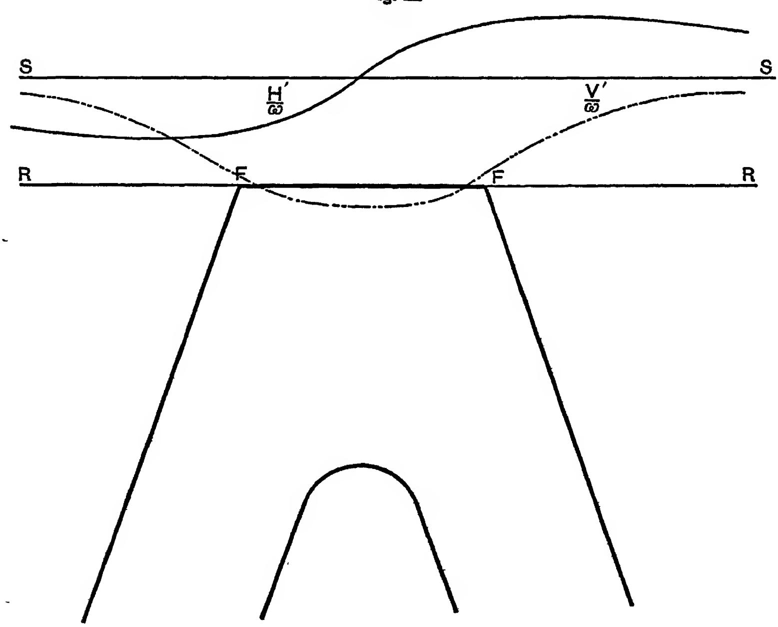


Fig. 21

the anticlinal is so truncated that magnetic material occupies the whole space on the rock-surface between the outer boundaries of the two formations. The angles of dip are equal, and are higher than in the preceding case, while the depth of covering is relatively much greater. The horizontal component is zero in the axial plane of the anticlinal, and has maximum values at two points, one on each side of the zero. The vertical component is a maximum at one point, also in the axial plane. The deflections produced by these conditions could not

Fig. 22



be distinguished in practice from those produced by a single vertically dipping formation.

In general, therefore, when two magnetic formations lie within range of each other's influence, the deflections are determined by the relative magnetic strengths of the two rocks, by their distance apart, by their strike and dip and by their depth of burial. It is evident that for certain given relations among these factors, the special cases above described will occur, and it is found that they really do occur in practice. For o er relations it is not possible to make a general statement ei .er

as to the number or the position of the maximum and minimum points.

Section VII.—The Interpretation of More Complex Structures.

The existence of two parallel belts of magnetic rocks may be accounted for geologically in more than one way. They may represent two distinct formations occurring at different horizons in the same series; or they may represent the same formation either duplicated by folding or separated into two parts, by the intrusion of a sheet of igneous rock parallel to its bedding. Since, then, two magnetic lines, the existence of which has been established by observation, may have more than one interpretation, the discrimination of these cases, when possible, is of special importance.

The question, whether any given case belongs to the first of these categories, can generally be settled only by following the lines of attraction into a district which affords a geological section across the formations involved, or by the occasional outcrop of the rocks which give rise to the disturbances, in which case lithological resemblances or differences, the relations to other formations, and the observed structure, will decide the matter one way or the other. In the special case, which of course can only rarely happen, in which either or both lines can be followed completely round an anticlinal dome, or a synclinal basin, the question would be settled affirmatively, even if outcrops were entirely lacking.

In the other instances the magnetic observations themselves often give means of discrimination, even when the outcrops are so few or so obscure as to be in themselves indecisive. It is characteristic of the folds in the pre-Cambrian rocks of this region that the axes are not usually parallel with the horizon for long distances, but are often inclined to it; in other words, when followed for greater or less distances, they pitch. The outcropping edges of any formation involved in an anticlinal or synclinal fold, which has been cut by a plane of denudation, will be parallel to each other wherever the axis of the fold is horizontal, but will approach each other where the axis is inclined. In an anticlinal fold they converge in the direction in which the axis sinks, while in a synclinal they converge in the

direction in which the axis rises. If the formation is a magnetic one, conformably placed between beds of non-magnetic character, the magnetic lines to which the outcropping edges give rise will therefore run parallel to each other when the axis is horizontal, and will converge or diverge when the axis pitches. The convergence or divergence takes place gradually, since the angles of pitch usually are not large.

In the case also of a single formation which stands on edge and has been split by the intrusion of a sheet of eruptive rock parallel to the bedding-planes, the magnetic observations will often show two parallel lines, which, at the extremities of the eruptive rock, where it wedges out, merge into one.

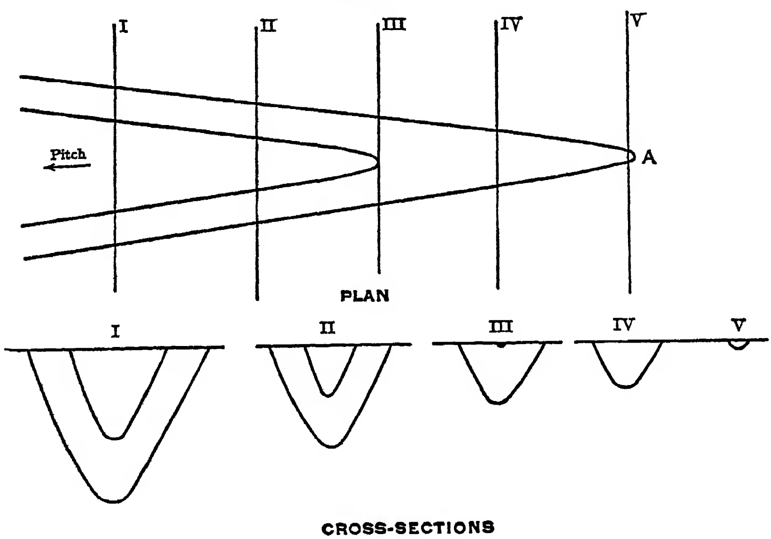
In general, therefore, two parallel magnetic lines which represent two distinct formations preserve their identity, and do not pass into each other; when, however, they represent the same formation, they will often come together if followed far enough. The principles which have already been applied to the analysis of simpler cases, are useful in discriminating among the three cases of convergence.

1. *Pitching Synclines.*—Let us first consider a pitching synclinal fold, which is represented in plan and by successive cross-sections in Fig. 23. It is evident that on the lines of traverse along sections I. and II. the deflections of the needle will observe the usual sequence for two parallel belts, the details depending upon separation and depth of covering; while on lines along sections III., IV. and V. the phenomena will be those caused by a single belt of magnetic rock. Also, on account of the rise in the axis, the south-poles of the rock are brought continually nearer the surface on these successive cross-sections, and therefore the two components of the rock-force will be smaller for each traverse than for the one preceding. Since the magnetic material comes to an end at A, it is no longer true that there is as much magnetic material on one side of these sections as on the other. Consequently the horizontal component due to the pull of the rock does not become zero at any point along these sections, but for every station has a positive numerical value and acts in the general direction in which the synclinal pitches. At the station in the plane of symmetry of the fold this component acts parallel to the axis. The direction and amount of the deflection depend upon the direction of strike and pitch of the synclinal.

Let us suppose first that the axis of the synclinal strikes north and pitches north. In this case Section I., in Fig. 23, is the most northern, Section V. the most southern.

The traverses along Sections I. and II. display the usual phenomena for two parallel belts. East of the eastern limb and west of the western the horizontal needle will be deflected towards the syncline. Between the two limbs there will be at least one point of no deflection, and frequently, depending upon the relations between the depth of burial and the thickness of

Fig. 23



the intervening non-magnetic material, either two other points of no deflection or two zones of retardation, one on each side of this middle zero.

Along Sections III., IV. and V. there will be but one point of no deflection of the horizontal needle, which will correspond with the axial line of the fold. Since this axis is north-and-south, and so coincides with the magnetic meridian, the horizontal component of the rock-force coincides in direction with the horizontal component of the earth's pull, and consequently there is no deflection of the horizontal needle. For other stations east and west of the central station the deflections are towards the west and east, with the usual maximum points.

The deflections on successive sections south grow smaller, since the angle between the two horizontal components progressively diminishes. The relative value of the horizontal component of the rock-force also progressively diminishes, since the thinning of the magnetic material due to the rise in the axis of the fold brings the buried north-poles into prominence. Therefore, the deflections of the horizontal needle, after the magnetic rock has been left behind, very soon become imperceptible.

The dip-needle deflections for the northern Sections I. and II. reach their maximum values at the usual points, over the central zero and near the outside zeros or points of retardation. For the southern sections the dips grow less, since the horizontal restoring-couple due to the rock has always a positive numerical value, and also because the vertical component of the rock-force diminishes, owing to the nearness of the south-poles. As the section approaches the limits of the magnetic material, the points of maximum dip become less and less clearly defined, and the dip-curve passes into an irregular line, slightly depressed below the line of no deflection. The reasons for this are, of course, obvious from what has been said above.

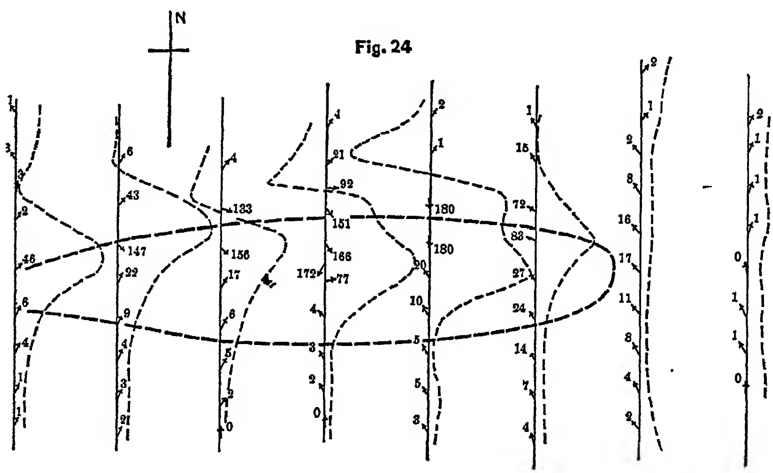
In the case of a synclinal fold pitching south, Section I. (Fig. 23) becomes the most southern, Section V. the most northern line of traverse. Sections I. and II. present the same general phenomena as before for both needles. In Sections III., IV. and V. the horizontal component due to the rock has a positive value for all stations as before, but in this case acts in a generally opposite direction to that of the horizontal component of the earth's force. Therefore, on these sections we should expect at first greater deflections of the horizontal needle, which would diminish rapidly as the sections approached and passed beyond the northern limit of the magnetic material, but which, for corresponding sections, would be greater than for the northerly pitching fold. The deflections of the dip-needle would also be greater for the same reasons.

For a synclinal pitching west, Section I. is the most western, Section V. the most eastern traverse. In this case, along I. and II., the deflections of the horizontal and dip-needles are dependent for their details upon the ratio of depth to distance of separation, but if far enough to the west will show clearly two

belts of magnetic material, approximately parallel, and striking approximately east-and-west. For Sections III., IV. and V., in which the distance of separation is either nothing or relatively small, the phenomena will indicate but one belt. On these sections, owing to the fact that the horizontal component of the rock-pull is nowhere zero, and has everywhere a general westerly direction, the deflection of the horizontal needle will be westerly throughout, and will reach a maximum north of the east-and-west axial plane of the material, where the angle which it makes with the magnetic meridian is more than 90°.

In accordance with the general principles stated in the discussion of a single belt with the same strike, the angles of dip

Fig. 24



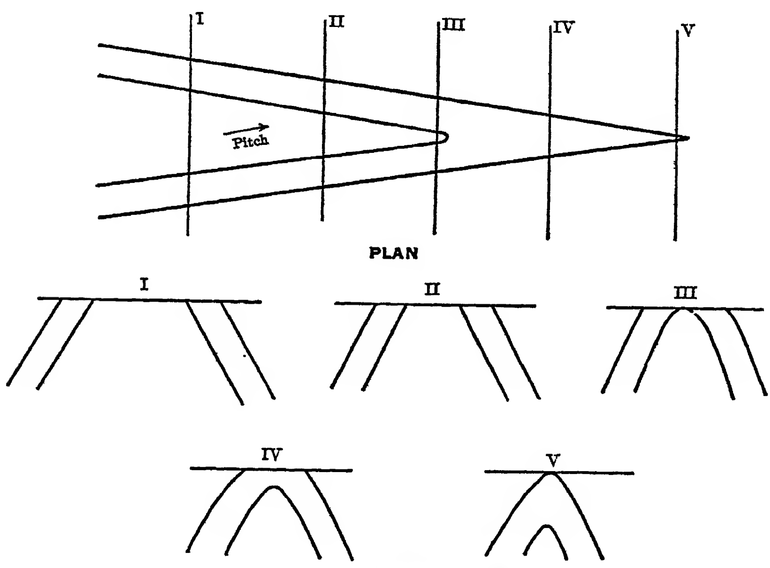
are in general smaller south of the syncline than north, and the maximum dip is reached at a point north of the axial plane. On sections farther to the east, near the limits of the rock, and beyond them, the dip-needle deflections, like those of the horizontal needle, rapidly diminish and soon become imperceptible. These facts are well shown in Fig. 24, which represents a series of north-and-south traverses across the Groveland basin, the limits of which are defined by outcrops on the eastern side.

In this figure it is instructive to notice the small dip-angles in the sections east of the end of the syncline. In the first of these the dip-curve shows a hollow near the axis of the fold, or angles of depression less than the normal. This is easily

understood upon considering that since the surface-covering is here small, the vertical component of the rock-force becomes very small at these stations compared with the horizontal component.

For an east-pitching syncline it is obvious that the facts will be entirely similar to those stated above, except that the deflections of the horizontal needle will be towards the east instead

Fig. 25



of towards the west. This is also well shown at the western end of the Groveland basin in Fig. 24. This basin does not show the phenomena of two lines, however, from the fact that it is so narrow and shallow that it does not include in its interior any over-lying non-magnetic material, and there is accordingly no separation of its rims.

2. *Pitching Anticlines.*—In the cases of pitching anticlines (Fig. 25) the sequence of observations in the area of separation of the rims is very similar to that of pitching synclines. In the zone of coincidence the structural difference in the two cases is that the material does not come to an end, but con-

tinues as one band, which, as the axis sinks, is progressively buried to a greater depth. Therefore, in general, the buried north-poles of the magnetic formation are not brought nearer the surface; and this, together with the fact that the material continues on in the line of the axis, produces characteristic phenomena in the magnetic sections.

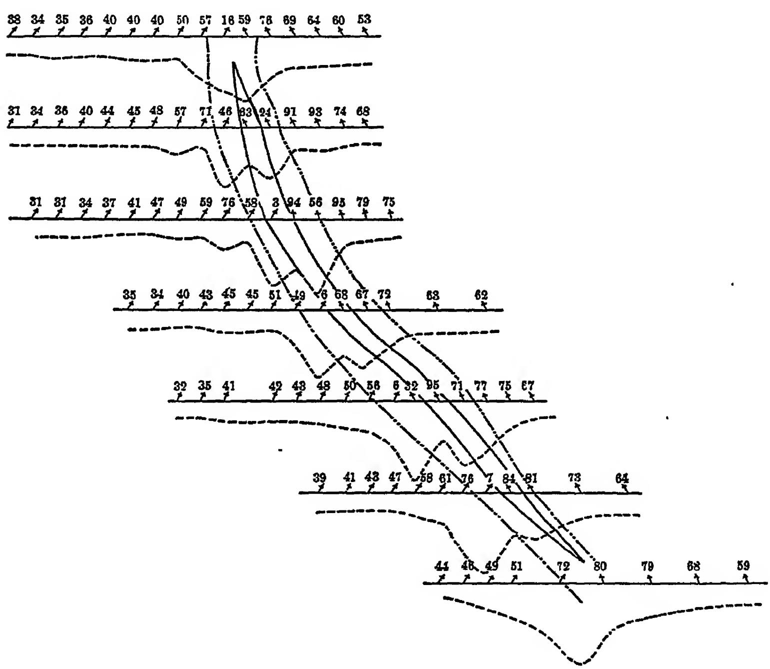
These phenomena, the details of which can be easily followed out for any given direction of pitch, and need not be described in detail, show in general two lines of attraction merging into one, which continues in the same direction as a strong line, showing, as it is followed, the peculiarities due to an increasing depth of burial. The points of maximum deflection of the horizontal needle continue to separate from each other on successive sections. The dip-curve shows a definite maximum closely corresponding, except for due east-and-west strikes, to the point of no horizontal deflection. Where the axis of the fold is so oriented that these points can be established they indicate the nature of the fold. If the strike is east-and-west, in which case they become indeterminate, the continuity of the line and its very gradual decrease in power frequently give an excellent basis for inference as to the nature of the fold.

3. *Formations Split by Intrusives.*—When a single formation has been split into two by the intrusion of a non-magnetic igneous rock, there are in the area occupied by the igneous rock two parallel magnetic formations, which give rise on cross-traverses to phenomena, the precise features of which depend upon the strike and dip of the formation, and upon the relation which the width of the intruded mass bears to the depth of burial. To describe these would involve a mere repetition of what has been said before. Such intruded masses always have a definite limit in length, which is usually not very great. When the limits are reached the two parallel lines pass into a single line which continues on in undiminished vigor. Also, such intruded masses are seldom confined to definite horizons for great distances and seldom split the formation into symmetrical halves. Two nearly parallel lines of unequal strength, which, as they are followed, become equal, and then again become unequal, with the stronger on the opposite side, are often, therefore, characteristic phenomena of this case. A good illustration of the unequal division of a magnetic rock

at different points along its strike by an intruded sheet, which wedges out at both ends, is given in Fig. 26. Between the second and third traverses from the north end the existence of this sheet has been proved by drilling.

4. *Summary.*—The means of discrimination among these cases of convergence are therefore founded on the deflections in the critical areas, where the separated bands of magnetic material merge into one. Strong deflections towards the point where

Fig. 26



they run together, with a rapid disappearance of all disturbances within a short distance of this point, indicate a pitching synclinal fold. A long continuance of the disturbances, with the characteristic phenomena attending deeper burial, beyond the point of coincidence, indicate an anticlinal fold. A coincidence in both directions and the continuation of the disturbances without diminution indicate an intrusive sheet. To these may be added the delicate criterion which the unsymmetrical distances of the

horizontal maxima from the central zero may afford. If, in the area of separation, the two belts depart from each so far as to be out of range of each other's influence, and it is found on successive cross-sections that the nearest maxima are inside the lines of no deflection which directly indicate the position of the rock, it can be concluded that the rocks dip towards each other, and, on the other hand, if the nearer maxima are outside these lines, that they dip away from each other. In the one case a syncline, and in the other an anticline, would be indicated, and, of course, in either case it would be certain that the phenomena could not be due to an intruded mass.

Section VIII.—Irregular Phenomena.

In what has gone before the cases described have presented clear-cut phenomena, which lead to definite conclusions. While, for the greater part of the area covered by our work, the character of the phenomena is very fairly represented by these cases, it would be misleading to leave the impression that all the observations in the whole series were of this character and that they presented no irregularities. As a matter of fact, irregularities did occur.

By irregularities is not necessarily meant change in the deflections of the instruments in successive traverses. That there should be no change in these it is necessary that there should be no change in strike and dip and depth of covering over the magnetic formations; in other words, that there should be a geological uniformity and no structural features on which light is necessary. By irregularity is meant those changes which in the light of present knowledge are not clearly related to other (chiefly structural) facts, and are departures from the course of the phenomena which follows from our fundamental conceptions and assumptions.

To illustrate: If our assumptions are absolutely and always expressive of the truth it would be impossible, on a traverse crossing a north-and-south-striking rock, for the horizontal needle to point south of the east and west line. Any exceptions to this rule would be irregularities. Such exceptions are exceedingly rare, yet they do occur. Thus, in the course of our work, some twenty traverses crossed magnetic rocks having a precise north-and-south strike, on which

about 200 observations of the horizontal needle were made close to the rock. Of these 200 observations a single one showed a deflection of the horizontal needle to the south of the normal. Again, in accordance with our assumptions, there should be a continuous progress in the pointings of the horizontal needle from zero up to the maximum and back again to zero in approaching a magnetic rock. Sometimes the observations do not show this regular march. To take still another example, parallel traverses across the same rock, when there has been, so far as known, no change in strike, dip and depth of covering ought to show precisely equal dip-curves and the same horizontal deflections at corresponding stations. In fact, close similarity is the rule, but absolute equality does not occur at all.

Many irregularities of the kind stated are no doubt to be ascribed to the fact that the assumption as to the uniform distribution of magnetite is never exactly true. While this assumption is the only one possible to make, in the lack of complete knowledge, for the purposes of analysis and illustration, it is only statistically true. It applies to the whole assemblage of facts better than any other, no doubt, but in certain extreme cases the departure of the actual phenomena from those expected may be of moment.

The larger part of the irregularities, however, are probably due to the fact that the rock-surface and the surface of the ground are not geometrical planes, and are not parallel, as has been assumed. The sub-aerial surface is really irregular and uneven, and the rock-surface must be of the same character. A local depression of the surface of the ground would carry a station located in it nearer the magnetic rock than it belonged by virtue of its horizontal position; a local elevation would carry it farther away. In both cases there would be a break in the regular sequence of the observations to be expected from theory. Likewise, the protrusion of a knob above the general level of the rock-surface would give this knob a disproportionate weight in the general resultant of the rock-force and interrupt the sequence at stations in the neighborhood.

The irregularities, therefore, while they show that certain assumptions, made for the purpose of bringing the facts within the reach of analytical treatment, are somewhat ideal, are yet

on the whole not subversive of the fundamental conceptions of the manner in which the magnetic force is contained in the rocks, but on the contrary decidedly strengthen them. One of our fundamental conceptions is that the south-poles of the little magnets are all at the rock-surface. An irregularity which could only be explained by supposing that the north-poles were at the rock-surface would be subversive of this conception. But no such case has been found among the observations at 10,000 stations.

Another source of anomalous deflections is no doubt to be found in the intrusives, which are usually abundant in the outcropping areas, and are unquestionably equally abundant in the covered areas. These, by locally increasing the width of the magnetic formations, or by entirely interrupting them, or by splitting them into many separated sheets, produce changes in the sequence of deflections which cannot clearly be ascribed to definite causes. Many apparent irregularities due to these causes would cease to be such, if observations were made at closer intervals, and the traverse-lines were nearer to each other.

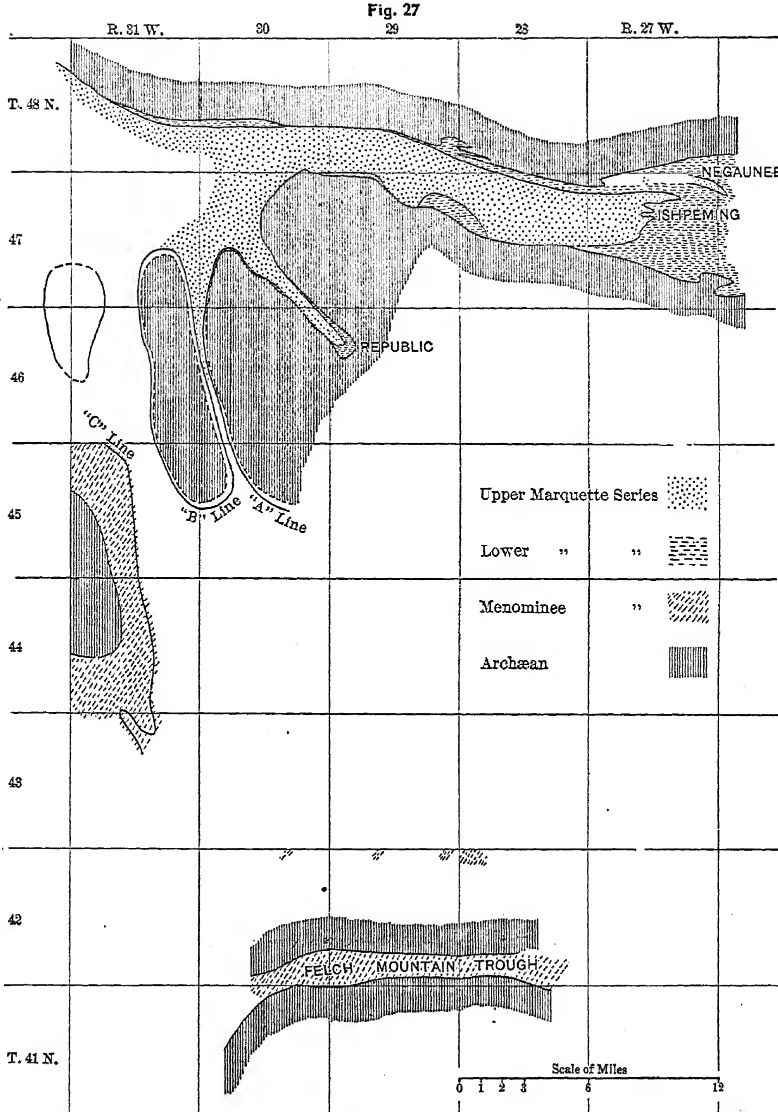
Section IX.—Description of the Results Reached in a Special Case.

As an example of the power of magnetic methods in unravelling the geology of a favorable region it may be of interest to give an account of our results and the steps by which they were reached in the area between the Marquette and Menominee iron-districts in Michigan. These results are not only of some importance in themselves, since they have added 25 miles of iron-formation, with its economic possibilities, to what was known before, but they threw the first clear light on the relations between the Marquette and Menominee rocks, which previously had been a matter of speculation only; and also have been instrumental in settling the question of the true place of certain of the Marquette rocks in the geological column. That these matters may be clearly understood, it is first necessary to give a short account of the geology of this portion of the Upper Peninsula.

The main belt of the Lower Menominee iron-bearing rocks, in which are situated all the productive mines of the Menominee range which lie east of the Ludington and Chapin, has a

present determined extent from Iron Mountain on the west to

Fig. 27



GEOLOGICAL MAP OF PART OF THE UPPER PENINSULA OF MICHIGAN

Compiled from the Maps of Brooks and Rominger, the Township Plats, and the Unpublished Notes of J. R. Finlay and H. L. Smyth

the vicinity of the Breen mine on the east, where it entirely disappears under the horizontal mantle of Palæozoics. Through

this distance it lies on the north side of the Menominee rivei which forms the boundary here between Michigan and Wis consin.

With our present knowledge the Lower Menominee serie may be broadly divided into four members. The lowest o these is a vitreous white or light-colored quartzite, usually ver heavily bedded, and certainly often many hundred feet in thick ness. This rests unconformably on the Archæan complex o gneisses, granites and schistose rocks of many kinds. Abov the quartzite is a white or cream-colored crystalline dolomite also many hundred feet thick. Next comes a slate formation, ii which are enclosed the great lenticular masses of iron-ore witl associated jasper, which have been so extensively worked a Iron Mountain and Norway. The thickness of the slates anc included ore on one section is somewhat less than 300 feet. Ii certain other places it is known to be less than this; and else where it may be more. Finally, the highest member of thi lower series certainly known to exist, is a lean jasper, consisting essentially of iron-ore and variously-colored quartz, some o which is fragmental, often banded or confusedly mixed, whicl has already been referred to as the Michigamme jasper.

When the range comes to be studied in detail, it may be found convenient further to subdivide these four formations, oi other additional members may be discovered.

The geological structure of the Menominee range has nevei been closely determined. There can be little doubt, however that it will be found to be a synclinorium probably complicated by extensive faults.

About 13 to 16 miles north of the Menominee river is the nearly parallel east-and-west-striking synclinorium of the Felch mountain range, in which these same four formations occur in the same order of superposition. Between the two ranges are at least two other pinched-in folds of undetermined extent from which erosion has removed for the most part all but the lower formations. The intervals between are occupied by the Archæan.

North of the Felch mountain range a few scattered outcrops of rocks which belong to one or another of these formations are known; but it is not until we reach the Michigamme and Fence rivers in Townships 43 and 44 N., Range 31 W., that

their relations have been determined. Here we find the upper three of the four formations, the jasper, the slate, and the dolomite, while the lower quartzite has disappeared as a quartzite, and is represented partly by an arenaceous slate, and also by a greatly increased thickness of the dolomite. These formations may be traced north up the Fence river for 12 miles. The only very important change of character consists in the mingling, and perhaps eventual total replacement, of the slates with rocks of volcanic origin.

This series along the Michigamme and Fence rivers is the eastern part of a ring of sediments and volcanics surrounding an oval area of Archæan granites and granitoid gneisses, from which they dip away in all directions.* Along the Fence river the direction of dip is easterly, at angles varying from 70° to as low as 20°. Outcrops are not very numerous; but they are sufficient to establish the fact that the Menominee formations, with the possible exception of the jasper on the west side, completely surround the crystalline core. In the case of the jasper the magnetic disturbances to which it gives rise permit its precise location along a continuous magnetic line from the southern end of the dome to its northern end, along a distance of 12 miles. In this distance it is brought to light less than half a dozen times in natural exposures and in test-pits; but the magnetic disturbances connecting these are so vigorous and clearly defined that there can be no possible doubt of its continuity.

This jasper is the Michigamme jasper, and the magnetic line which establishes its position may be called the "C" line.

We have then in T's 44 and 45 N., R. 31 W., three of the Lower Menominee formations overlying a dome of Archæan crystalline rocks, and dipping towards the east.

In the productive portion of the Marquette range between Negaunee on the east, and Republic on the west, the Lower Marquette series consists of two or three clearly-marked formations, which, perhaps, may further be subdived according to individual taste. The lowest of these, which rests on the Archæan complex, is fragmental in origin, and is prevailingly a white vitreous quartzite, which in one or two localities is conglomeratic near the base. Often it is represented by a muscovite

* The eastern half only of this Archæan oval is shown in Fig. 27.

schist as the result of the dynamic metamorphism of the original arkose. In the eastern part of the productive area, and along the northern side of the main fold in the western part of the district, this formation is overlain by siliceous slates. Elsewhere the slates are not present, or are not known.

The next higher formation is the Negaunee iron-formation, which has already been described in Section II. This rock, which has many phases, as there noted, is clearly marked off from the lower quartzite by its great richness in iron, and by the fact that over the whole Marquette district it nowhere appears to contain fragmental material, except in the transitional zone between it and the lower formations.

Above these conformable formations comes the unconformably-placed Upper Marquette series, the base of which rests, now on one member, now on the other, or on the Archæan.

East and south of Negaunee, and extending thence to the shore of Lake Superior at Marquette, is a series of rocks which resemble lithologically neither the Upper nor the Lower Marquette series. It consists, in ascending order, of quartzite, with basal conglomerates, dolomite, and slates. This series, named by Wadsworth* the Mesnard series, has been regarded by him, as well as by Van Hise,† as belonging with the Upper Marquette series, or at least as overlying the Lower Marquette formations just described. Major T. B. Brooks had earlier correlated the dolomite with the Lower Marquette quartzite, and had supposed that there was a gradual‡ passage from one into the other along the strike.

This series is found only on the extreme eastern edge of the Marquette area, between Goose Lake and Lake Superior, over a length of about 6 miles. Elsewhere, over by far the greater part of the Marquette range, no member of it has been recognized.

The geological structure of the Marquette range presents the general features of an east-and-west-striking complex syncline or synclinorium. The pre-Cambrian sedimentary rocks, with their associated instrusive and extrusive igneous rocks, occupy the trough, in which there is much local complexity of structure,

* *Report State Board Geological Survey, Michigan*, 1893, p. 65.

† *Bull. Geol. Soc. Am.*, vol. v., 1893, p. 5.

‡ *Geol. Sur. Mich.*, vol. i., pp. 108, 109.

and are flanked on the north and south by the older Archæan crystallines.

At the western end of the range (Fig. 27) the peculiar Republic trough branches from the main synclinorium, and runs southeast into the Archæan rocks for 6 or 7 miles, having a nearly constant width of about half to three-quarters of a mile. In this trough the Algonkian rocks have been so closely compressed that they stand essentially on edge. The interior is occupied by the younger Upper Marquette quartzites and schists, between which and the underlying Archæan walls the older Lower Marquette iron-formation and quartzite here and there occur.

The northwestern end of the Republic trough is about the western limit of mining development, though not of exploration, on the south side of the Marquette synclinorium. Up to this point outcrops, producing mines, and old explorations are sufficiently abundant to permit the separate formations to be traced and mapped with comparative ease, and to indicate at least the larger structural features.

At this northwestern end of the Republic trough the lower Marquette series makes an abrupt turn to the south, and may be followed for a mile or more by occasional outcrops and test-pits. The Negaunee iron-formation is persistently present beneath the Upper Marquette quartzite, and gives rise to a very strong and persistent line of attraction, which was followed in our work for about 12 miles to the south and southeast. For about 4 miles from the sharp turn at the mouth of the Republic trough, it runs nearly due south; afterwards it turns somewhat to the east of south, and follows that course for about 6 miles, after which it turns more and more towards the east, and finally, where we left it, its course was only slightly south of east. That this magnetic line is caused by, and marks the position of the Negaunee iron-formation, there cannot be the slightest doubt, for that rock outcrops in a few scattered localities, occurs abundantly in the drift, and has been found in occasional test-pits and drill-holes throughout this distance. The underlying quartzite outcrops beneath the iron-bearing formation near the northern end of the line; but farther south it is entirely covered by the drift, so far as the territory has been examined. The overlying Upper Marquette rocks are also known to exist just west of the

Negaunee formation for a large part of this distance. The magnetic line which accompanies the Negaunee formation may be called the "A" line. Taking into account the connected Republic trough and its exposures of the Lower Marquette rocks, it is seen that the "A" line partially surrounds a dome of the Archæan crystallines, and that in going from the interior of this dome outward across the "A" line we pass from older to younger rocks. The dip along the "A" line is, therefore, on the whole, towards the west, although the observed dips at the few localities where determinations have been made are either vertical or slightly inclined towards the east.

The southern end of the "A" line, as far as it has been traced up to the present time, passes through Sections 5, 8, 9, 15 and 16 of T. 45 N., R. 30 W. In Section 5 it is just 5 miles east of the "C" line, which, as has already been said, is caused by the Michigamme jasper, a magnetic rock occupying a definite place in the Menominee series, which again is underlain by other typical Menominee formations, and finally by the Archæan.

Between the "A" and "C" lines is a third magnetic line, which may be called the "B" line. This was traced parallel to the "A" line and less than half a mile away, from near the south end of the latter to the north end, and finally entirely round an elliptical area, closing again upon itself at the starting-point, the perimeter of the ellipse being 25 miles in length. Throughout this entire distance not a single outcrop could be discovered along the "B" line. Within the enclosed area, however, several exposures of granites and crystalline schists were found, which left no doubt that the greater part of the area enclosed by the "B" line was occupied by Archæan rocks of the same general character as those partially enclosed by the "A" line on the east and entirely enclosed by the "C" line on the west. The area between the "A" and "B" lines was also proved to contain the basal member of the Upper Marquette series. The southwestern quadrant of the "B" line ellipse is nearly parallel to the "C" line and only a mile and a half away.

The known facts with reference to the "B" line are these: (1) It represents a magnetic rock; (2) this magnetic rock completely encircles an Archæan core. It can further be inferred with

practical certainty that this magnetic rock, which carries such constant magnetic properties for 25 miles, must be sedimentary. With regard to its structure, the foregoing considerations would necessarily involve the conclusion that it dips away from the Archæan core in all directions, and this conclusion is fortified by the unsymmetrical separation of the horizontal maxima on the magnetic cross-sections. It follows, therefore, that on the eastern side of the oval, where it is parallel to the "A" line, it dips towards the east, and on the western side, where it is parallel to the "C" line, it dips toward the west. This conclusion is further supported by the dips within the ellipse in the outcropping Archæan rocks that show structure. These all happen to lie east of the major axis, and all dip towards the east.

East of the "B" line, and between it and the "A" line, is found the basal member of the Upper Marquette series. The rock which is manifested in the "B" line must, therefore, be older than any member of the Upper Marquette series. The Negaunee iron-formation, represented in the "A" line, dips west, while the rock of the "B" line dips east. They are both older than the basal member of the Upper Marquette series, and are both younger than the Archæan. They are both strongly and persistently magnetic. For 8 or 10 miles they run parallel to each other less than half a mile apart. Their broad structural relations to the Archæan basement of the region are precisely similar. Therefore, although the rock that gives rise to the "B" line has never yet been seen, it may be concluded with the utmost confidence that it is the Negaunee iron-formation, and that the "A" and "B" lines represent this rock brought up in the two limbs of a narrow and probably deep synclinal fold.

This conclusion carries the Negaunee iron-formation 3½ miles farther to the west, and, in the northeast part of T. 45 N., R. 31 W., leaves a gap of but 1½ miles between the Lower Marquette and the Lower Menominee series.

Here, between the "B" and "C" lines, is precisely the same situation as between the "A" and "B." One magnetic rock represented by the "B" line dips west, the other, the Michigamme jasper, represented by the "C" line, dips east. Between them no magnetic disturbances can be found. The area

between them must have a synclinal structure, and if they are not one and the same formation, then each must undergo an extremely rapid and precisely similar change in lithological character (*i.e.*, the loss of magnetite) in a very short distance, and be represented on the opposite side of the synclinal fold by a non-magnetic formation. Each of these rocks is persistently magnetic in the direction of the strike for great distances. That each should independently lose its magnetite in the direction of the dip in this particular locality is very improbable. And so the grounds for the conclusion that the "B" and "C" lines represent one and the same rock are quite as firm as those upon which rests the conclusion that the "A" and "B" lines represent the same formation.

Therefore, depending entirely upon the structural results of magnetic work and upon the lithological inferences that may at the same time be drawn from it, the conclusion has been reached that the Negaunee iron-formation of the Marquette district is identical in age with the Michigamme jasper of the Menominee district. Furthermore, this conclusion which is, as geological results go, of an exceptionally high order of probability, has been arrived at by working over a district in which outcrops are essentially non-existent.

This conclusion once established has important consequences upon our ideas of the early geological history of what is now the Upper Peninsula of Michigan. First, however, it should be said that there are excellent reasons for believing that the Michigamme jasper represents not only the Negaunee iron-formation, but also the quartzite which immediately underlies it over the southern half of the Marquette range as well. These reasons need not be stated here, as they could not be made clear without too long a description of the results of petrographical study.

The first consequence is that the formations which constitute the whole of the Lower Marquette series over the 25 miles or more of the productive and best-known portion of the range, are represented in the Menominee district and the intervening area by a single formation, and that the highest in the Lower Menominee succession—the Michigamme jasper. The Menominee formations below the Michigamme jasper are then all older than the Marquette rocks and do not occur at all within the

area of the Marquette rocks proper. This suggested at once two questions: (1) why these lower formations are absent from the Marquette area; and (2) whether the Mesnard series, in the eastern marginal area, which lithologically and in the order of superposition of the individual members is remarkably like these Lower Menominee formations, may not be really below and not above the lower Marquette series.*

Additional field-study has answered the second question in the affirmative, and the Mesnard series has since been shown by the United States Geological Survey really to underlie the wide-spread formations of the Lower Marquette series proper.

To the first question there seem to be two answers which are *a priori* possible. It is conceivable that the Menominee quartzite, dolomite and slate, or some of them, were deposited in a succession of unbroken sheets over the whole Marquette area, in continuity with the similar Mesnard formations, and that afterwards the main Marquette area was elevated above the sea and entirely stripped of these formations by long-continued denudation. Finally, when the time of deposition of the Michigamme jasper came round, this elevated area had again been reduced to sea-level and subsided below it, so that the Negaunee iron-formation and the Michigamme jasper were deposited in an unbroken sheet over the whole. If this hypothesis is correct, two consequences should follow from it. First, we ought to find some discordance between the Michigamme jasper or the Lower Marquette quartzite and the lower formations in the marginal areas between the Menominee and Marquette and Mesnard and Marquette areas respectively, or at least a gradual cutting out of these lower formations by the iron-bearing members and the lower quartzite. And second, we ought to find, in the lack of discordance, rocks present in the areas of continuous deposition, which represent the time of denudation.

With regard to the first of these consequences no verification is possible, at least in the territory between the Marquette and Menominee districts, from the lack of outcrops. From the northwestern end of the Republic trough in T. 47 N., R. 30 W., to the "C" line in T. 45 N., R. 31 W., there are no exposures what-

* *Am. Jour. Sci.*, vol. xlvii., March, 1894, p. 222.

ever of the Algonkian rocks which underlie the jasper. In this distance of about 11 miles there is certainly room for a cutting out of the lower formations. Nor (for the same reason) can it be definitely settled whether elsewhere farther within the Menominee area there is any discordance. That there is general parallelism between the jasper and the lower rocks, and strict conformity in some places, is true. But this is not at all inconsistent with a period of erosion between them, if that erosion antedated the later and more severe orogenic disturbance. In at least one place there is a very great structural discordance visible, which may very likely, however, be due to faulting.

What the situation is in the Mesnard area I do not know, but the facts at present known on the Menominee side may be said not to exclude the possibility of some period of erosion in the Menominee area below the jasper.

With regard to the second consequence, viz., the deposition in the submerged areas of formations which would represent the erosion-period in the elevated area, the evidence at hand is decidedly against the existence of such formations.

The alternative hypothesis is that the lower quartzite, dolomite and slate formations of the Menominee and Mesnard areas were never deposited over the Marquette area at all, and this hypothesis is much more likely to be the true one. We can suppose, as I have already pointed out,* that this part of the Upper Peninsula was a slowly subsiding area, the central portion of which, now occupied by the Marquette rocks, stood at a greater elevation above the encroaching sea than the rest. While the quartzite-dolomite-slate triad, pointing to deepening water, was going down in the Mesnard area on the east and the Menominee area on the south and west, the central Marquette area remained above the sea. At last, when the Michigamme jasper began to be deposited in the Menominee area, the Marquette high-land was finally submerged, and covered, as the sea marched over it, first with a sheet of arkose made up of its own disintegrated *débris*, and finally with the same non-elastic sediments as chiefly compose the Michigamme jasper.

To this very definite reading of the earlier geological history

* *Am. Jour. Sci.*, vol. xlvii., March, 1894, p. 222.

of this portion of the Upper Peninsula we are directly led as a consequence of the identification of the Negaunee iron-formation with the Michigamme jasper, through a magnetic survey conducted as set forth in this article. With this identification firmly established, the general relations of the Marquette and Menominee series become clear, although there is still room for doubt on certain points.

The Cyanide-Process in the United States.

BY GEORGE A. PACKARD, BOSTON, MASS.

(Colorado Meeting, September, 1896.)

When, in April, 1892, the writer began experimenting with the cyanide-process, it had already proved a success in the treatment of tailings, but had not become an important factor as a primary method of ore-treatment. The Livingston, Colorado, mill was running a few small agitators; the Mercur mill, in Utah, was said to be a success, but was "closed down to increase capacity;" and Mr. A. B. Paul, in Shasta county, California, was running ore through a stamp-mill into leaching-vats, using a weak cyanide solution in the battery. Mills had been, or were being, erected in Arizona; on the Comstock lode; and in South Dakota. Some of these never started, and several were shut down or remodelled for other processes. Altogether, the outlook was anything but favorable.

Many of these mills were planned to make use of agitation, and a cyanide solution containing at least 1 per cent. of potassium cyanide was employed. Since then the improvement of the process, both chemically and mechanically, has placed it among the recognized successful methods of ore-treatment.

Within the past year I have visited a large number of cyanide-mills, and have collected considerable data, of which I publish enough herewith to give an idea of the development of the process and of the methods followed in the principal mills. I would say, however, that nearly all of these plants are experimenting, most of them employing experienced chemists,

and are so constantly making improvements, increasing extraction and decreasing cost of treatment, that the figures here given are only approximate.

Table I. shows the character and approximate composition of the ore treated by cyanide at a number of mills, and Table II. gives the details of treatment (pages 719 and 720).

The process has been applied on a large scale only to rather low-grade, highly siliceous ores, containing but a small percentage of base metals and having their value principally in gold. In fact, the cyanide-process has a field of its own. I have been told of one instance in which cyanide competed successfully with the smelters on ore carrying as high as 4 ounces in gold, the ore being one in which the value was easily extracted to a high percentage. In the Cripple Creek district, where an extraction of 90 per cent. is obtained in from four to six days, and where the smelting-charges were from $5 to $7 per ton, ore running as high as $40 was in 1895 bought by the cyanide mills.

With silver-ores, while some very good results have been obtained, the length of time required for treatment has usually been too long, and the consumption of cyanide too high, for the process to give economical results. There are, however, several plants in the vicinity of Tombstone, Arizona, working on silver-ores. In the case of ores containing from 1 to 10 ounces of silver, in addition to a commercial gold-value, the process has been advantageously employed. Thus the Golden Reward Company, in South Dakota, having certain ores containing from 1 to 5 ounces of silver which was lost in chlorination, has built an addition to the plant, in which such ores are treated with cyanide.

Chlorination is the only process the field of which the cyanide method is seriously invading. For mines located at a considerable distance from a railroad the cost of transportation of the chemicals used in chlorination has been hitherto an almost prohibitory factor; and here, at least until the use of liquid chlorine becomes a practical success, cyanide has the advantage. At the Golden Reward plant, early in 1895, they were using for chlorination about 35 pounds of chemicals per ton of ore, while only 2½ pounds were necessary for treating 1 ton with cyanide. As already observed, if there is silver present, the

cyanide has the advantage that part of the silver is recovered; but, so far as my observation goes, the gold-extraction is usually higher by chlorination than by cyanide.

With amalgamation, cyanide enters into competition only in the case of very finely-divided gold, which is saved more or less successfully in pans. Generally speaking, ores suited to one process are not suited to the other. Thus at the mill at Cooke, Montana, there was a sudden change last year in the character of the ore. A quantity of very fine free gold appeared in a mine where a "color" had but rarely been seen before. This gold, although so fine as to pass through a 60-mesh screen, could not be extracted by the strength of the solution employed, and in the length of time usually allowed on the ore there being treated, while the value of this new ore was too low to permit the use of a stronger solution (entailing larger cyanide-consumption) or a longer time. To save this gold, and a little pyrites, the tails were passed over a system of riffles and blankets. A similar arrangement has since been used at one of the Florence, Colorado, mills, to save the coarse gold resulting from roasting ores containing tellurides.

The treatment of low-grade concentrates has not generally proved a success, on account of the long time and high consumption of cyanide involved; and smelter-rates are too favorable to make the process profitable on high-grade concentrates, save in exceptional cases.

There are at least two districts in this country where the cyanide process has proved a panacea: Camp Floyd (Mercur), Utah, and Gilt Edge, Mont. To these Cripple Creek might be added; for the ability to sell low-grade ores has certainly aided largely in the development of that camp.

At Mercur, Mr. E. A. Schneider* says "the gold occurs as a fine coating on particles of magnetic iron" in limestone. The ore is somewhat porous, and the value occurs largely along certain lines of cleavage which become the lines of fracture, enabling the ore to be treated with only a very coarse crushing. In fact, at one mill, the Marion, treating fifty tons a day, the only crushing-machinery is a No. 2 Gates crusher, from which the ore goes to a $\frac{1}{8}$-inch-mesh trommel, the oversize being re-

* *Eng. and. Min. Jour.*, May 18, 1895.

turned. In nearly all of the mines in this district some cinnabar and more or less arsenic occurs; much of it, however, is deposited on the hanging-wall in such a manner that very little is sent to the mills. At the Marion mill, I was told that they sometimes retorted the "slimes" on cleaning up, in order to save the quicksilver deposited on the zinc. At the Geyser mill I was told that they sometimes found arsenic as the oxide, though never enough to amount to more than ½ per cent. All of the arsenic which I saw in the district was a sulphide. It has been found that the presence of a very small quantity of arsenic in the ores soon fouls the solutions so that they cannot be titrated for standardizing; and at most of the mills the solution is thrown away if it becomes badly fouled.

The Gilt Edge ore contains porphyry and limestone, and is not unlike the ores of the Camp Floyd district. There are but two mills in operation here.

The ores of the Cripple Creek district are a porphyry (or phonolite), containing a fraction of 1 per cent. of tellurium, some of which, as Prof. Anderson, of the American Reduction Co., informs me, is combined with gold and some with iron. When the tellurium occurs in the oxidized condition, the ores are easily treated raw; but in the case of other ores a preliminary roast is required. Up to the present time the glut of ore at Cripple Creek has been so great that the cyanide-plants have been able to discriminate closely in buying, and have avoided ores which offered either chemical or mechanical difficulties. I am told that the roasting of ores in this district is done largely for mechanical reasons, as the ore leaches much more freely after roasting.

In South Dakota, the ores all contain a little sulphur, and in some the sulphur runs as high as 5 per cent. Both mills have found that a higher extraction is obtainable after roasting, but it is accompanied by a higher consumption of cyanide.

A plant at Plumas was operated for a time on concentrates, but was abandoned, being unable to compete with the smelter.

In general, ores containing much sulphur have given unsatisfactory results. I obtained some very good extractions in tests of small quantities of pyritic ores at Cooke, Mont., using cyanogen bromide, and Mr. H. C. Cutler obtained similar results at the University of Minnesota. Mr. Wallace Macgregor, in

charge of the Congress, Ariz., cyanide-mill, reports a 93 per cent. extraction from ore containing pyrites, which had received but a slight roast, and which he treated with potassium cyanide alone. Mr. Macgregor, in some recent experiments on ores containing a small percentage of pyrites, found that where the ore received a thorough roast, the consumption of cyanide is decreased; and a similar effect has been noticed after roasting at the Florence mill of the American Reduction Co.

The size of ore leached varies from through a ⅞-inch-mesh at the Marion to through a 40-mesh at the Metallic Reduction Co.'s mill at Florence; and the crushing- and sizing-machinery ranges from the single Gates crusher and one trommel at the former, to three Gates crushers, three multiple-jaw crushers, six sets of rolls and numerous trommels at the latter.

The method of conveying ore from pulp-bin to leaching-vats is almost universally by cars on a track over the vats. The Commercial mill, at Bingham, Utah, has inclined spouts, through which the ore flows from a centrally located bin into the tanks, and a few mills have bins directly over the tanks.

The vats are usually round, and vary from 9 to 26 feet in diameter and from 2 to 5 feet in depth. A few mills have all-steel tanks. In Utah many of the vats have sides of iron and wooden bottoms—a very satisfactory arrangement. Other mills have vats of pine, cypress or redwood, sometimes unpainted, but usually painted with paraffine- or asphaltum-paint.

The false bottoms consist of a frame of strips, or of boards in which 1-inch holes are made, covered with jute or matting. Over this a No. 8 duck is sometimes used, especially if the tank is to be emptied by sluicing. In a few mills a gravel filter is used.

The method of leaching I find to be quite variable. The preliminary treatment includes the use of lime, caustic soda and sodium dioxide. Lime, when used, is mixed directly with the ore. In some cases the pulp is then washed with water until the lime is all washed out. At other mills the solution is put on at once. Caustic soda is used in the same way, and also in solution as a preliminary wash. At the Commercial mill, Bingham, Utah, Mr. Stephens told me he had found that a preliminary treatment with a solution of sodium dioxide in water gave better results than either lime or caustic soda, and considerably decreased the time required for leaching.

Many mills begin the leach by admitting the solution at the bottom of the vat, until the ore is covered. The solution is then turned on top and allowed to run on at the top and drain off at the bottom simultaneously for a certain number of hours, the surface of the ore being kept covered. Very good results are obtained from this mode of treatment when the tanks are allowed to stand for a short time, after the pulp is covered, before the drainage-valve is opened and the solution turned on top. This allows the whole mass to become thoroughly saturated and the slow draining prevents the formation of channels. Any great depth of solution on top of the charge causes it to "pack," and an uneven extraction follows. In the Mercur district the ore is covered with solution, which is allowed to stand from thirty minutes to six hours and then drawn off. This operation is repeated from eight to thirty-five times. Here the material leached is so coarse that there is no danger of "packing." Each operation of covering takes from two to six hours. A few mills cover the pulp with solution, allow it to stand forty-eight to ninety-six hours, draw it off and wash. Many of the mills follow the strong solution with a wash of weak solution ($\frac{1}{10}$ per cent. or less). This is in turn followed by a water-wash, which flows through the zinc-boxes into the weak-solution tank, and becomes the first wash for the next charge.

A few mills warm the solution, and at one mill I found facilities for steaming the charge before putting on the solution.

The use of sodium dioxide in connection with cyanide, which is known as the Kendall process, has been adopted at a few mills. Prof. Kendall says that the better results are obtained by adding the freshly-made solution of dioxide to the cyanide solution; but many chemists claim that much oxygen is lost in this way, which is made available when the dioxide is mixed, dry, with the ore. The object of the dioxide is to furnish "nascent oxygen," which "acts on the cyanide to liberate cyanogen," hastening the reaction. My own experience with the use of dioxide has been that, in general, a greater extraction is obtained in a short time (twenty-four to forty-eight hours); but that at the end of a long time (seventy-two to one hundred and twenty hours) the extraction with dioxide is about the same as with cyanide alone.

The only method of removing the gold from the solution known to the writer to be in practical use in this country is by

precipitation on zinc shavings.* An attempt in Arizona to precipitate electrically on lead sheets (not the Siemens-Halske method, no iron anodes being used) proved a failure. The same plant is now experimenting with precipitation by "zinc-fume."

Fig. 1 shows the form of zinc-box in common use. The box shown here is made of 2-inch plank, dressed and bolted together, and painted with paraffine-paint. It has six compartments 13 by 20 inches in size and 20 inches deep.

The screen on which the fine shavings rest is 4 inches above the bottom. In the bottom of each compartment is a 1-inch pipe, closed by a stop-cock, through which the slimes are drawn off in cleaning up. The zinc-boxes at the Cripple Creek mill have these discharge-pipes at the bottom, through the side, and discharge into a trough leading to a tank. At the Mercur mill long sheet-iron boxes are used, having wooden partitions wedged in place. These are easily removed for cleaning-up, and the slimes are all brushed together.

Fig. 2 shows the form of the boxes at the South Dakota mills. No iron is used in their construction and they are carefully painted with asphaltum. A series of these is used in lieu of a larger box divided into compartments. They have the advantage that they are small and easily handled in cleaning up.

About sixty pounds of zinc shavings are necessary to fill a box of the size shown in Fig. 1, and this will precipitate the gold from about 1500 pounds of 0.2 per cent. solution per hour, the solution carrying from 0.1 ounce to 0.8 ounce of gold per ton on entering the zinc-box, and from 0.01 to 0.05 ounce on leaving it. The gold in wash-waters and weaker solutions is less easily precipitated, a much longer contact with the zinc being required.

On cleaning up, the zinc is washed, the slimes are screened through a sieve varying from ¼-inch-mesh to 60-mesh at different mills, and the coarse stuff is returned to the zinc-box. In this country, where slimes are treated at the mills they are subjected to the action of acid (usually sulphuric), the zinc is thoroughly washed out and the residues are fluxed and melted.

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* Electrical precipitation is said to be in use in the Okanogan district, Washington. See *Min. and Sc. Press*, Oct. 17, 1896.

A few mills ship the slimes to smelters or refiners, but the difficulty of obtaining a satisfactory sample and the almost con-

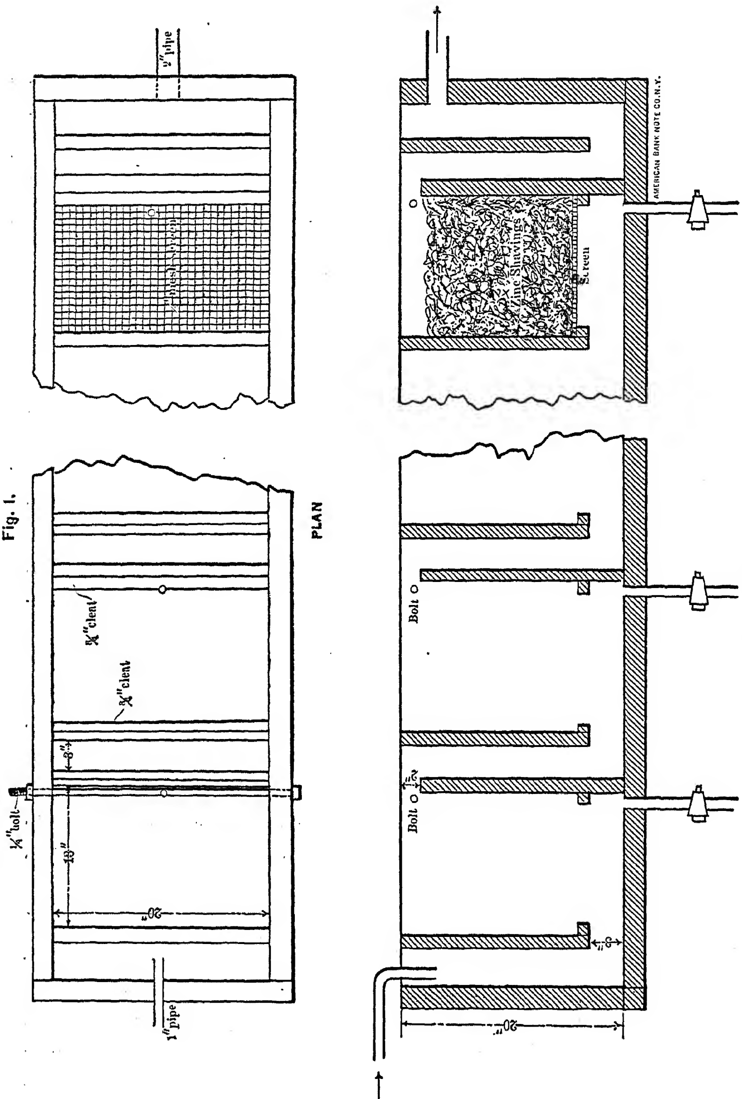


Fig. 1.

PLAN

SECTION THROUGH CENTER

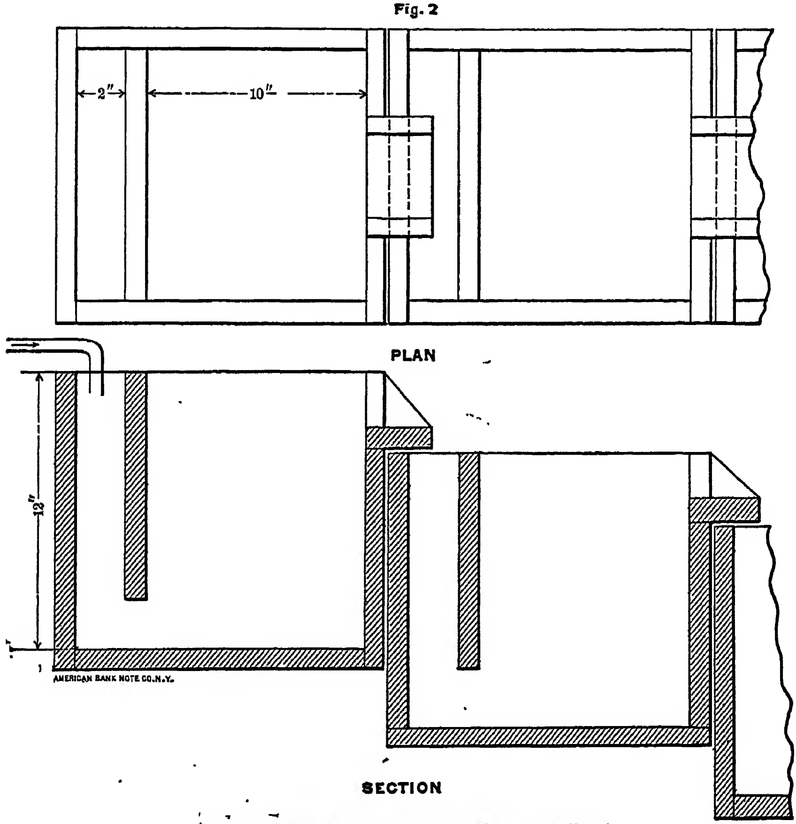
ZINC BOXES AT COOKE, MONTANA.

stant wide disagreement between buyer and seller have led many smelters to refuse to handle them.

At the Gilt Edge, Montana, mill, an attempt was made to chlorinate the slimes, with only partial success, it being found necessary to roast and melt the residues in order to obtain all the gold.

The consumption of cyanide varies with the character of

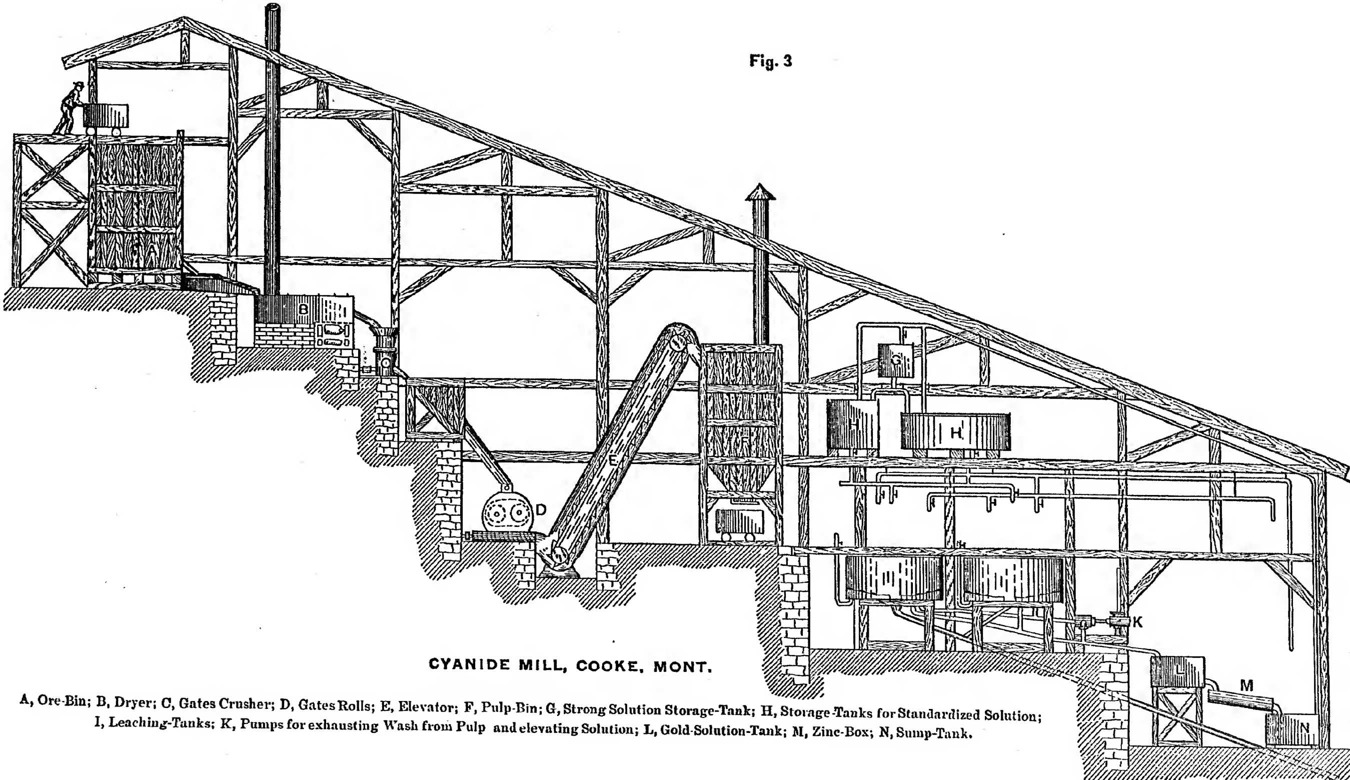
Fig. 2



ZINC BOXES AT DEADWOOD, SOUTH DAKOTA

the ore and depends very little on the amount of gold and silver extracted. It varies from 2 pounds per ton, reached occasionally on Cripple Creek ores to $\frac{2}{10}$ pound on the Gilt Edge ore. The latter ore is an ideal one for the process, showing no indications of decomposed pyrites and the resulting acid compounds. Acid ores, although a preliminary treatment be given to neutralize the acid, consume more cyanide than others.

Fig. 3



CYANIDE MILL, COOKE, MONT.

A, Ore-Bin; B, Dryer; C, Gates Crusher; D, Gates Rolls; E, Elevator; F, Pulp-Bin; G, Strong Solution Storage-Tank; H, Storage-Tanks for Standardized Solution; I, Leaching-Tanks; K, Pumps for exhausting Wash from Pulp and elevating Solution; L, Gold-Solution-Tank; M, Zinc-Box; N, Sump-Tank.

The consumption of cyanide in the zinc-boxes varies with the strength of the solution, the length of time it is in contact with the zinc, and the amount of other salts in solution. Using a 1 per cent. cyanide solution, and having mixed with the ore in the tanks an excess of impure lime containing alumina and magnesia, I have had as high as 3 pounds of cyanide per ton of solution consumed in the zinc-boxes. Ordinarily, with a solution of 0.2 per cent. KCy entering the boxes, the consumption is practically nothing. The strength rarely falls below 0.17 per cent. KCy on leaving, and often shows no perceptible loss. A series of experiments made by the writer last winter indicated that the loss of cyanide in the zinc-box is less when lime is used than when the acid in the ore is neutralized with soda, the excess of lime or soda and resulting salts not being washed out before the cyanide solution is added. The consumption of zinc, however, was slightly less when soda was used. This latter result was confirmed by Mr. Macgregor at the Congress mill.

TABLE I.—*Character and Composition of Ores Treated in the United States by the Cyanide Process.*

MILL.	CHARACTER OF ORE.	COMPOSITION.					
		SiO_2.	Fe.	Al_2O_3.	$CaCO_3$.	S.	Other Elements.
		Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	
Mercur, Utah...	Porphyry and Limestone..........	Similar	to Sun	shine...	....	....	As, Hg.
Sunshine, " ...	"Decomposed porphyry and quartz"..	70 to 80	....	5 to 10	12 to 15	Little.	As, Sb, Te.
Bingham, " ...	"Sandy quartz with iron oxidized from sulphides and bunches of lead carbonate"...........	....	....	....	Little.	....	As, 0.2 per cent. Cu, much free acid.
Gilt Edge, Mont.	Porphyry and Limestone.						
Cooke, "	Decomposed porphyry..................	84	3.6	2.6	Ca 0.6	0.3	
Black Hills, S.D.	"Silica, with iron in varying conditions"................	80 to 95	Varies.	....	....	Varies.	Traces of Cu, Mn, As, Sb.
Cripple Creek, Colo........	Porphyry..............	....	....	....	CaO 1 to 3	Small.	Te.

Fig. 3 shows the arrangement of the mill at Cooke, Montana.

The cost of "cyaniding" varies largely with the character of the ore. There are a number of mills which crush and "cyanide" ore for less than $2 a ton, exclusive of royalty paid to the

TABLE II.—*Details of the Cyanide Process as Practiced in the United States.*

MILL.	Capacity. Tons per day.	Size of Ore Leached. Mesh.	Preliminary Treatment.	Strength of Cyanide Solutions.	Number of Hours Leached.	Cyanide Consumed per Ton of Ore Treated.	Zinc Consumed per Ton of Ore Treated.	ASSAY OF ORE. Gold.	ASSAY OF ORE. Silver.	EXTRACTION. Gold.	EXTRACTION. Silver.	Cost per Ton.	AUTHORITY.
		Inches.		Per cent.		Pounds.	Pounds.	Oz. per ton.	Oz. per ton.	Per cent.	Per cent.		
Mercur..............	200	3/4	None.	0.20 to 0.25	48	.60	....	.5 to 6.	...	80 to 87	...	$0.85	G. H. Dern.
Sunshine...........	60	1/4	Lime.	0.06 to 0.20	48	.75	.30	......	...	....	...	......	C. H. Jacobs.
Bingham...........	40	1/4	Sodium Dioxide.	0.025 to 0.125	120	1.0	.30	.25	2.0	80	...	1.85	W.A. Stephens.
Gilt Edge.........	50	20	None.	0.15	72	.2	.25	.45	...	75	...	1.25	H. S. Sherard.
Cooke..............	25	16–20	Lime.	0.25	48	1.5 to 2	.25	Varia	ble.	....	...	......	Private notes.
Black Hills........	40	30	Lime.	0.50 to 0.75	96	1.25	....	.3 to .9	...	85	...	2.50	I. S. Childs and notes.
Cripple Creek.....	75	40	Various, Lime.	0.05 to 0.75	100	1 to 2	.25 to .50	1	0.2	90	...	......	B. Hunt.

Ore at Bingham and Black Hills has to be dried before crushing.
Gilt Edge cost includes hauling 1 mile from mine.

company owning the patents. The lowest cost I have heard of is 85 cents a ton, at the Mercur. No company has yet been able to reduce the cost of treating tailings to the minimum reached in South Africa, 59 cents per ton; but one plant operating under exceptionally favorable conditions is working at a cost of 69 cents a ton. In general, the tailing-plants working in this country do not obtain a high extraction.

There are a large number of tailing-plants in the United States, especially in the southwest, where the hot, dry climate renders expensive buildings and drying-machinery unnecessary. Including the output of these mills, I find that nearly 200,000 tons of ore and tailings were treated by cyanide in 1895, producing over $1,000,000 in bullion value.

Laboratory-Tests in Connection with the Extraction of Gold from Ores by the Cyanide Process.

BY H. VAN F. FURMAN, DENVER, COLO.

(Colorado Meeting, September, 1896.)

As the cyanide-method for the extraction of gold from ores is extensively used in the United States and elsewhere, and appears destined to prove a factor of increasing importance in the metallurgy of gold, a description of the latest laboratory-methods may prove of interest.

The history of the development of the process in the United States has been analogous to that of all new processes. Many failures are to be recorded and a few successes. It is the opinion of the writer that, had the following simple tests been better understood, many of the failures would not have occurred, and we should probably have a larger number of successful plants in operation. While the process is not simple, but requires a high degree of chemical and engineering skill, the determination of the adaptability of an ore to the method is generally not difficult. While the laboratory-results will not always coincide with the actual results obtained in the mill, they will serve as a guide and control on the working of the mill, and will generally suffice to determine if the ore can be

economically treated by the process. Objection may reasonably be made that in small tests conditions different from those which occur on a larger scale are introduced. However, tests made on from 25 to 100 pounds of ore should be closely duplicated in the mill. Tests on a smaller scale will serve to show what may be expected in the mill.

In determining the adaptability of an ore to this method of treatment, and the percentage of extraction which may be expected in the mill, the following must receive consideration:

The percentage of extraction will generally be somewhat higher in the laboratory than in the mill, but the consumption of cyanide will also be higher in the laboratory.

A most important point is the character of the ore, particularly the manner in which the gold is contained in it; whether it is in the free state; coarse or fine; intimately associated with pyrites or other sulphides; the character of the sulphides with which it is associated; whether it is combined or alloyed with bismuth, tellurium or other elements. An examination by the eye, with or without the aid of a magnifying-glass, will frequently settle these points. Should the sample be in a finely pulverized condition, careful vanning may be resorted to with advantage.

The size to which it is advantageous to crush the ore will depend largely upon the character of the ore and its gangue. Should the gold be unevenly disseminated throughout the material, and the gangue be hard and non-porous, fine crushing is essential to a good extraction. The Cripple Creek ores are examples of such material, crushing to 40-mesh, or finer, being necessary to insure a good extraction of the gold. Some ores, however, being very porous, readily permit the solutions to percolate, and consequently can be successfully leached in a coarse condition. The Mercur ores are examples of such material, pieces one-half inch or more in diameter being successfully leached.

Whether the ore requires roasting prior to leaching will depend principally upon the condition of the gold in it. If present as a telluride, the gold may be extracted from the raw ore by fine grinding and leaching with potassium cyanide; but the action of the solution on the tellurides is slow, and it may be

more economical to subject the ore to a preliminary roast. Roasting presents two disadvantages: it materially increases the cost of treatment, and it is liable to result in the formation of salts detrimental to the subsequent leaching; for example, soluble sulphates, which decompose potassium cyanide. Unless the roasting is carefully conducted, gold may be lost by volatilization. This is especially the case with tellurides. On the other hand, many ores are rendered more porous by roasting, and thus the rate of percolation is increased and coarser particles can be leached.

The best strength of solution and the time of maceration and percolation are important points. The time required and the amount of cyanide consumed will depend largely upon the strength of the solution employed. For example, the time required with a 1 per cent. solution will generally be much less than with an 0.25 per cent. solution. On the other hand, for many ores the consumption of cyanide with a 1 per cent. solution is so great that the process becomes commercially impracticable, while with dilute solutions the ore may possibly be treated with success. In this connection, mention may be made of the use of additional reagents, such as sodium peroxide, bromine, etc., which *are said* to facilitate the extraction in many cases. This is a mooted point, and the use of these reagents is objected to by some, as they have a tendency to increase largely the consumption of zinc, when zinc is the precipitant employed, and to foul the solution. According to the equation of Elsner,

$$4Au + 8KCN + O_2 + 2H_2O = 4KAu(CN)_2 + 4KOH,$$

oxygen, or an oxidizer, is essential to the solution of the gold. It would appear to the writer that the necessary oxygen can be obtained from the air at less cost, and quite as conveniently, as by the introduction of expensive reagents.

The amount of potassium cyanide required per ton of ore, including the amount which is destroyed during treatment, is a vital point. The character of the ore and its associated minerals will determine the consumption of cyanide.

The rate of percolation of the solution through the ore is a question of some importance. It would appear as if too much

had been made of this point, as it is essential that the solution should remain in contact with the ore for a considerable time in order to insure a good extraction. On most ores, the percolation will generally be quite as rapid as the necessary time of contact will permit.

A small quantity of the ore should be ground to 30-mesh, which is a suitable degree of fineness for the preliminary experiments. A sample is carefully cut out of this prepared pulp, ground to pass 100-mesh, and assayed for gold and silver in the usual manner.

A quantity of stock-solution, containing 0.5 or 0.6 per cent. of potassium cyanide, should be prepared. As this solution is liable to decomposition, it should be kept in a stoppered bottle, protected from the air and sunlight, and should be tested from time to time according to Test 6 (page 730).

It is necessary that the water used in making up the stock-solution should be quite pure. It should always be tested for impurities, for should it contain iron salts, magnesium sulphate, salts with an acid reaction, soluble sulphides, free carbonic acid, or sulphuric acid, they will decompose the potassium cyanide.

1. Determination of Acidity.

Should an ore be acid, the result will be decomposition of potassium cyanide unless this acidity is destroyed before the cyanide solution is added.

Soluble Acidity.—Agitate 10 grammes of the pulp for 10 minutes with 50 c.c. of water; filter, and test the filtrate with litmus-paper for acidity. Should acidity be shown, wash the ore until the washings no longer give an acid reaction when tested with litmus-paper. Now titrate the total filtrate with deci-normal caustic soda-solution, until the neutral point is obtained, using litmus as an indicator.

Latent Acidity.—Transfer the washed ore to a small porcelain evaporating-dish; cover with water; add a measured excess of deci-normal caustic soda-solution; stir and titrate the excess of soda with deci-normal acid-solution. This gives the latent acidity.

Total Acidity.—The sum of the above tests gives the total acidity, but as this is frequently all that is required, it may be determined as follows: Introduce 10 grammes of the pulp into

a stoppered bottle with some water; add a measured excess of the caustic soda-solution, agitate for 20 minutes and then titrate back with deci-normal acid-solution.

The soluble acidity is due to salts with an acid reaction, such as ferrous sulphate, zinc sulphate, copper sulphate, etc., or to free sulphuric acid from the decomposition of pyrites, tellurous acid, etc. It may be overcome by giving the ore a preliminary wash with water. This washing is followed by treatment with a weak solution of caustic soda or caustic lime, which neutralizes the latent acidity due to basic salts. The amount of alkali necessary is determined from the quantity of deci-normal soda-solution used in the above experiments. Unless the ore contains a large amount of free acid, the preliminary washing with water may be omitted; the total acidity being determined and reported in terms of lime. Sufficient lime is then added to the ore, before crushing, so that it becomes thoroughly incorporated before the ore reaches the tanks, and when the tank is charged, the cyanide solution is at once admitted.

2. Test for the Consumption of Cyanide.

The original strength of the stock-solution being known, it is only necessary to determine its strength after it has been used on a lot of ore, to arrive at the consumption.

Introduce 20 grammes of ore (treated with a sufficient quantity of soda or lime, if necessary) into a glass-stoppered bottle; add 40 c.c. of the cyanide solution, and agitate for 20 minutes; filter; measure off 20 c.c. of the filtrate and determine the amount of undecomposed potassium cyanide remaining in the solution, according to Test 6. The difference between the amount of potassium cyanide in 20 c.c. of the stock-solution and the quantity found above gives the amount consumed by 10 grammes of ore.

If the consumption of cyanide is not excessive (which will depend altogether on the value of the ore, as rich ores can stand a much higher consumption than those of lower grade), or, say, not over 4 pounds of potassium cyanide per ton of ore, the following extraction-tests can be proceeded with.

3. Tests for the Percentage of Extraction.

Generally two series of tests are made: By agitation and by percolation.

Agitation.—Take four 4-ounce wide-mouthed glass-stoppered bottles, place 1 assay-ton of pulp in each, then add to each 60 c.c. of solution of the following strength respectively:

To No.	1,	0.1 per cent. KCN.
"	2,	0.3 " "
"	3,	0.5 " "
"	4,	0.75 " "

Before adding the cyanide solution, the proper quantity of neutralizer, as determined in Test 1, is added. Allow the bottles to stand for 48 hours, with occasional shaking. Filter off solutions; wash with water up to original bulk; test an aliquot portion of the solution for loss of cyanide; dry the tailings; crush them to 100-mesh, and assay. From the assay of the original pulp and the assay of the tailings, the percentage of extraction can be calculated.

Another method preferred by some is to assay the tailings and also assay the solution (see Test 11); then

$$\text{Percentage of Extraction} = \frac{\text{Assay of Solution} \times \dfrac{\text{Weight of Solution}}{\text{Weight of Pulp}} \times 100}{\text{Assay of Solution} \times \dfrac{\text{Weight of Solution}}{\text{Weight of Pulp}} + \text{Assay of Tailings.}}$$

For quick results, the bottles are placed in an agitator which is revolved for twenty-four hours.

Several agitation-tests can be carried out in this manner, varying the quantity of cyanide solution used and the mesh of ore (10-, 20-, 30- and 40-mesh).

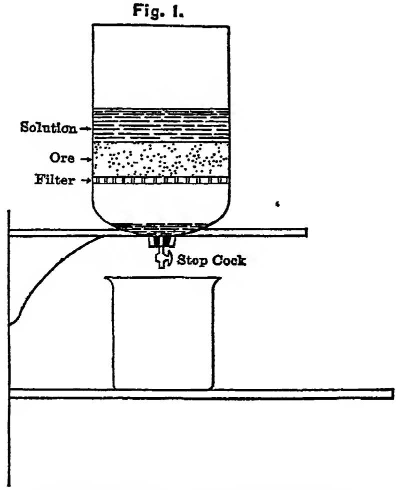
Percolation.—For these tests a glass percolating-jar provided with a false bottom covered with a double filter paper will be found convenient. Such an apparatus is shown in Fig. 1. Place 1 pound, or more, of the pulp (with the proper quantity of the neutralizer thoroughly mixed with it) on the filter, and add to the charge 230 c.c. of the stock-solution for each pound of ore taken. Allow the solution to macerate for twelve hours, and then percolate gently for thirty to forty hours. Wash with water until the filtrate reaches the original bulk. The rate of percolation may be noted here.

Test the solution for loss of cyanide; assay an aliquot por-

tion of the solution and the tailings, and thus determine the percentage of extraction.

A series of tests may be carried out in this manner, varying the strength of the cyanide solution, the fineness of the ore, and the time of contact from twelve to seventy-two hours.

Fig. 1.



Apparatus for Test by Percolation.

The results of these experiments will prove the applicability of the process to the ore in question, and the best method of treatment, *i.e.*, the strength of solution and the mesh which will give the best extraction in the shortest time, with the least consumption of cyanide.

Where it is desired to treat larger quantities of ore, a very convenient apparatus is a large glazed earthenware jar, provided with a false bottom and an outlet at one of the lower sides.

The following formula will be found convenient for calculating the percentage of extraction:

Let A = The assay-value of the ore in ounces Troy per ton of 2000 pounds Av.

B = Milligrammes of gold found in the filtrate.

C = Pounds Av. of ore taken for treatment.

X = The percentage of extraction.

$$X = 6.43016 \frac{B}{AC}.$$

4. DETERMINATION OF THE CAUSE OF CYANIDE-CONSUMPTION.

Should the consumption of cyanide be high, the cause of consumption may be determined by an analysis of the cyanide solution. For every part of cyanide rendered inoperative, a corresponding proportion of metal enters solution. Thus one part by weight of iron consumes seven parts by weight of potassium cyanide, etc. The following equations represent the reactions most frequently encountered:

$$FeSO_4 + 6KCN = K_4Fe(CN)_6 + K_2SO_4;$$

or

56 (at. weight of Fe): 390 (mol. wt. 6KCN) :: 1 : 7.

$$ZnSO_4 + 4KCN = K_2Zn(CN)_4 + K_2SO_4;$$

or

65 (at. wt. Zn) : 260 (mol. wt. 4KCN) :: 1 : 4.

$$2\,Cu''SO_4 + 6KCN = K_2Cu'(CN)_4 + 2K_2SO_4 + 2CN.$$

Salts of aluminium and magnesium act in a different manner with potassium cyanide, their hydrates being formed with the liberation of hydrocyanic acid, thus;

$$Al_2(SO_4)_3 + 6KCN + 6H_2O = Al_2(OH)_6 + 3K_2SO_4 + 6HCN.$$
$$MgSO_4 + 2KCN + 2H_2O = Mg(OH)_2 + K_2SO_4 + 2HCN.$$

A preliminary alkaline treatment overcomes this objectionable feature, their hydrates being precipitated, which are then inert towards potassium cyanide, thus:

$$MgSO_4 + Ca(OH)_2 = Mg(OH)_2 + CaSO_4$$

insoluble magnesium hydrate and insoluble calcium sulphate being formed.

Soluble sulphides, formed by the action of potassium cyanide on some metallic sulphides, again react to some extent on the cyanide, with the formation of sulpho-cyanide of potassium, thus:

$$ZnS + 4KCN = K_2Zn(CN)_4 + K_2S;$$
$$K_2S + KCN + H_2O + O = 2KOH + KCNS.$$

To determine the cause of the consumption of cyanide place 100 grammes of the pulp in a wide-mouthed bottle, add 200 c.c. of the cyanide solution and agitate for fifteen hours. Filter, take 20 c.c. of the filtrate (equivalent to 10 grammes of ore) and

evaporate almost to dryness in a porcelain dish. Add some strong sulphuric acid, evaporate almost to dryness, and cool. Dilute with water, add some hydrochloric acid, and heat to effect solution if necessary. The metal in solution may now be determined by the usual methods.

The strong sulphuric acid at a high temperature decomposes the metallic cyanides, thus:

$$2AgCN + 3H_2SO_4 + 4H_2O = Ag_2SO_4 + 2NH_4HSO_4 + 2CO_2 + 4H.$$

Strong nitric acid may be used in place of strong sulphuric; but hydrochloric cannot be used, as it leaves the metal in the form of a double cyanide salt, which is soluble.

The reactions with nitric and hydrochloric acids are:

$$AgCN + HNO_3 + 2H_2O = AgNO_3 + CO_2 + NH_3 + 2H.$$
$$K_4Fe(CN)_3 + 4HCl = H_4Fe(CN)_6 + 4KCl.$$

5. DETERMINATION OF THE CAUSE OF NON-EXTRACTION.

Should the above tests show a low percentage of extraction, the next step is to determine the cause of this non-extraction. It may be due to numerous causes, such as total destruction of potassium cyanide by certain salts of the base metals present in a form readily attacked by the potassium cyanide. The gold may be in a very coarse state, in which case the solvent action of the potassium cyanide will be too slow for the practical application of the process. The gold may be combined or alloyed with tellurium, antimony, bismuth, etc., in which case the cyanide is inoperative until the combination is broken up. The presence of soluble sulphides in solution. The character of the gangue, such as kaolin or talc, which may be present in such quantities as to effectually prevent percolation. To overcome these difficulties the following methods may be tried:

In the case of an ore which consumes a large quantity of cyanide, if a preliminary wash with water, weak acid or alkali is ineffective, the ore may be classed as one not adapted to the process.

The coarse gold difficulty may be overcome by amalgamation, either before or after treatment with cyanide, which gen-

erally results in an excellent extraction. The South African practice may be cited as an example.

The difficulty due to the presence of bismuth, antimony, etc., in combination or as an alloy with the gold, may sometimes be overcome by fine grinding and long contact with the cyanide solution; but the usual method is to treat the ore to a preliminary roast, which converts the gold into a condition in which it is readily attacked by cyanide.

The difficulty due to the presence of soluble sulphides can be overcome by the addition of a soluble lead salt or the addition of an oxidizing agent.

Should the ore contain much kaolin or talc, if coarse crushing is ineffectual, nothing further can be done, and the ore must be classed as one not adapted to the process.

Ores containing considerable quantities of oxidized copper minerals are to be classed as not adapted to the process.

6. Determination of the Free Potassium Cyanide in Solution.

A number of methods have been proposed, and there are several which give good results when the solution is free from cyanides other than potassium cyanide. With complex mill solutions, containing $K_2Zn(CN)_4$, these methods fail.

With pure solutions, such as the freshly-prepared stock-solution, a rapid and accurate determination may be made by titrating a measured quantity of the solution to be tested with a standard solution of silver nitrate, using 2 or 3 drops of a 5 per cent. solution of potassium iodide as an indicator. Silver cyanide is formed, and immediately redissolves in the excess of potassium cyanide. The reaction is as follows:

$$AgNO_3 + KCN = AgCN + KNO_3;$$
$$AgCN + KCN = KAg(CN)_2;$$

or the reaction may be expressed as follows:

$$2KCN + AgNO_3 = KAg(CN)_2 + KNO_3.$$

As soon as all the potassium cyanide has been converted into the double cyanide of potassium and silver an additional drop of the silver nitrate solution produces a pale yellow opales-

cence, owing to the formation of silver iodide. The quantity of silver solution added is read off from the burette and the percentage of potassium cyanide is calculated.

In the United States a solution containing 6.535 grammes of silver nitrate to the liter is used. The quantity of cyanide solution usually taken is 10 c.c. When 10 c.c. of cyanide solution are taken, each c.c. of the silver solution used indicates 1 pound of potassium cyanide to the ton (2000 pounds) of solution. Abroad, deci-normal silver nitrate solution is generally employed, the loss in cyanide being obtained in percentage on the ore. This percentage multiplied by 20 gives the pounds of potassium cyanide consumed per ton.

Another method which answers all purposes, provided the cyanide solutions are quite pure, but which is useless for complex mill-solutions, depends upon the following reaction:

$$KCN + 2I = KI + ICN.$$

This method requires a solution of pure iodine in potassium iodide. A solution of pure wheat starch is used as an indicator. Ten or more c.c. of the cyanide solution are measured off and run into a beaker. A few drops of the starch solution are added, and then the iodine solution is run in from a burette, with stirring, until an excess of one drop of the iodine solution is present, which is indicated by the formation of permanent blue iodide of starch. The iodine solution may be standardized by some freshly prepared stock-solution, or preferably by means of a standard solution of sodium hyposulphite. This method may be used to determine the percentage of KCN in commercial potassium cyanide.

A number of other methods for the determination of the available cyanide have been proposed, but that first described is believed to be the best.

In a well-regulated mill the strength of the solutions is tested on each tank every four hours whilst the solutions are percolating.

The method usually adopted for the determination of the free potassium cyanide in mill-solutions is as follows: 10 c.c. of solution are diluted with distilled water to 65 or 70 c.c. and titrated with the standard silver nitrate solution, without the use of an indicator. When all the free potassium cyanide is changed to

$KAg(CN)_2$ an additional drop of the silver solution produces a distinct white opalescence, owing to the formation of insoluble zinc cyanide, thus:

$$K_2Zn(CN)_4 + AgNO_3 = KAg(CN)_2 + Zn(CN)_2 + KNO_3.$$

It is probable that this second reaction commences before all the potassium cyanide is combined with silver; for on partly titrating a solution containing both salts, and allowing it to stand, a white precipitate slowly forms. Hence the titration must be performed rapidly, in which case the separation of the white precipitate can be taken as marking the end-point. The titration requires some practice to be performed properly.

7. Determination of the Free Hydrocyanic Acid in Solution.

To 10 c.c. of the mill-solution add 10 c.c. of a solution of potassium bicarbonate (containing 15 grammes of $KHCO_3$ to the liter), dilute to 65 or 70 c.c. and titrate as in Test 6, without the use of potassium iodide as an indicator. Upon the addition of the bicarbonate the following reaction takes place:

$$HCN + KHCO_3 = KCN + CO_2 + H_2O.$$

The titration gives the HCN in terms of KCN, and KCN $\times$ 0.415 = HCN. As this titration gives the potassium cyanide and the hydrocyanic acid, the difference between this result and that obtained in Test 6, multiplied by 0.415, gives the hydrocyanic acid.

For each c.c. of mill-solution taken 1 c.c. of the potassium bicarbonate solution is used. This will be sufficient for solutions containing as much as 0.4 per cent. of HCN, which is much higher than mill-solutions usually run, but the excess does no harm.

The addition of the bicarbonate solution usually causes a distinct turbidity, which should disappear when the solution is diluted, giving a clear liquid for titration. If, as rarely happens, a faint turbidity remains, a duplicate of the solution to be titrated is prepared, the end-point being shown by the increased cloudiness in the titrated solution as compared with the blank solution.

8. Determination of the Total Simple Cyanides in Solution.

To 10 c.c. of the mill-solution add 10 c.c. of half-normal sodium hydrate solution (20 grammes of NaOH per liter), dilute to 65 or 70 c.c.; add a few drops of the potassium iodide solution, and titrate to pale yellow opalescence, as in Test 6. The result is the total KCN, HCN and $K_2Zn(CN)_4$, in terms of KCN. The amount of sodium hydrate to be added depends principally on the percentage of $K_2Zn(CN)_4$ present, as a large excess should be avoided. The amount given will be sufficient for solutions containing 0.7 per cent. zinc and 0.4 per cent. hydrocyanic acid, and will answer in all ordinary cases likely to be encountered in mill-practice. The addition of the sodium hydrate produces a permanent precipitate, but the use of potassium iodide as an indicator prevents any doubt as to the end-reaction.

9. Determination of the Ferro-, the Ferri- and the Sulpho-Cyanides in Solution.

The ferrocyanides and the sulphocyanides, if desired, may be determined by titration with a standard solution of potassium permanganate in an acid solution, the reactions being as follows:

$$10K_4FeCy_6 + K_2Mn_2O_8 + 8H_2SO_4 = 10K_3FeCy_6 + 6K_2SO_4 + 2MnSO_4 + 8H_2O.$$

$$10KCyS + 6K_2Mn_2O_8 + 13H_2SO_4 = 11K_2SO_4 + 12MnSO_4 + 10HCy + 8H_2O.$$

One portion, acidified with sulphuric acid, is titrated, the result representing both of the above compound cyanides. To a second portion, acidified with sulphuric acid, a solution of ferric chloride is added. The resulting prussian blue is filtered off and the filtrate is titrated with the standard permanganate solution. This second titration gives the potassium sulphocyanide.

The permanganate solution should be quite dilute, containing not more than from 0.3 to 0.5 gramme of potassium permanganate to the liter. It may be standardized by any of the approved methods, and its value may be calculated for the compound cyanides according to the above equations.

Ferricyanide, if present, may be determined by reducing it to ferrocyanide, and then by titration with standard potassium permanganate as above.

Should sulphides be present, the shaking up of the solution with moist lead carbonate will produce a black precipitate of lead sulphide. When present, they must be thus removed, by agitation with lead carbonate and filtering off the resulting lead sulphide, before the compound cyanides can be determined.

10. Determination of the Zinc and Lime in Solution.

As these are sometimes of considerable importance, they have to be occasionally determined in the mill-solutions. The solution is treated in the manner described in Test 4, and the lime and zinc can then be determined by conventional methods.

11. Determination of the Gold and Silver in Solution.

Numerous methods have been proposed, but the following which is the simplest, is the method generally adopted:

Evaporate one assay-ton (= 29.2 c.c.) of the solution to dryness in a lead tray. Roll up the lead and cupel on a hot cupel, weighing the resulting button. Alloy the bead with silver, if necessary, and part for gold as usual.

Should the solution contain over 0.2 ounce of gold per ton, the lead should be scorified together with a little borax glass prior to cupellation.

The lead tray is made of pure lead foil, and is 3 inches long, 2 inches wide and ½ inch deep. It should weigh about 20 grammes. For the evaporation, the tray containing the solution is placed on a piece of asbestos card-board, heated by a burner underneath.

In conclusion, the author wishes to express his indebtedness to Mr. Philip Argall, manager, and Mr. L. G. Eakins, chemist, of the Metallic Extraction Company, and to Mr. William Orr, chemist of the Gold and Silver Extraction Company of America, for valuable notes which have been largely used in the preparation of this paper. Acknowledgment is also due to the courtesy of Messrs. John Wiley and Sons, who have kindly permitted the use of considerable material contained in the fourth edition of the author's "Manual of Practical Assaying."

The Solution and Precipitation of the Cyanide of Gold.

BY S. B. CHRISTY, PROFESSOR OF MINING AND METALLURGY, UNIVERSITY OF CALIFORNIA, BERKELEY, CAL.

(Colorado Meeting, September, 1896.)

THE fact that many millions of gold have been extracted by the cyanide process, during the last five or six years, from South African tailings which could not be profitably worked by any other method previously tried upon them, lends a peculiar practical interest to this branch of metallurgy.

Numerous writers have recently made valuable contributions on the process without exhausting the many complexities of the subject. Among the first American writers on the subject in its modern aspect were two former students of the University of California, Mr. Louis Janin, Jr., and Mr. Charles Butters. Young Janin was one of the first to enter this field, and performed much useful experimental work in the early stages of the history of the process.

At about the same time, Mr. Charles Butters, who had left California to erect chlorination works for the Robinson mine at Johannesburg, became interested in the cyanide process, and, after getting the chlorination plant in working order, put up, for working the tailings from the Robinson mine, a cyanide plant which achieved the first large-scale success ever made with this process.

The papers which Charles Butters, together with John E. Clennell and Edgar Smart, have contributed on the chemistry of the process, were among the first to give any adequate idea of its nature.*

Another important contribution to the literature of the subject is the paper of Dr. A. Scheidell, published in 1894 as a "Bulletin of the California State Mining Bureau." This contains an admirable summary of the state of the art at the time of its publication.

* *Eng. and Min. Journal*, Oct. 8, 22, 29, 1892; Nov. 2, 1895.

Many notable contributions have been made by others, some of which will be referred to in what follows, as called for by the context.

The rapid extension of this process in South Africa is evidenced by the value of the cyanide-product for the Witwatersrand district. In 1890 it was less than $6000; in 1891 it had grown to over $60,000; in 1892 to over $3,000,000; in 1893 to over $6,000,000; and it is still increasing.

Although, several years before 1891, I had obtained some very promising extractions from gold-ores by the use of cyanide of potassium solutions, I was prevented until that year, by the constant pressure of other duties, from any systematic study of the subject. In the experimental work then commenced I was aided by several of my students, more especially by Mr. Thomas E. Eichbaum, who at the time of his death, a couple of years later, was chemist for Mr. Almarin Paul at his cyanide-works in Shasta county, Cal.; Mr. C. W. Merrill, who has since erected cyanide-plants at Bodie, Cal., and at Harquahala in Arizona; Mr. Leslie Simpson, now in South Africa; Mr. H. C. Baldwin, now in Mexico; and Mr. F. Booth, now in Alaska.

The results of these studies, made on a large variety of ores, was to show in all cases a partial extraction of gold. This extraction varied from 20 per cent. to 95 per cent. While it was always easy to make a partial extraction of the gold, it proved difficult, and in many cases impossible, to get anything like a complete extraction. Another difficulty encountered in these experiments was that very dissimilar results were often obtained with ores of almost exactly the same composition.

A careful study of all this preliminary work soon convinced me that two equally important classes of causes were active in bringing about these variations in the result: the one of a chemical, the other of a mechanical, nature. It was equally clear that it was expedient to make a separate study of these two sets of causes; and as it was of course impossible to separate them in the treatment of samples of ores, a systematic investigation was laid out, involving the use of simple artificial products of known composition, so as to determine the various complicated elements of the problem.

In the long, involved and difficult investigation which followed, and which has occupied all my leisure time since 1892,

I have been greatly aided by my assistant, Mr. E. A. Hersam, for whose intelligent skill and faithful devotion to the work I wish to express my warm appreciation.

The great complexity of the chemistry of the cyanides, the lack of accurate information in the chemical text-books, and the consequent erroneous statements in many of the publications on the cyanide process, have greatly added to the difficulties of the study. It became necessary to test *de novo* almost every step of the ground, so that an investigation which I had hoped to finish in a couple of years has already taken twice that time, and is now only fairly begun. Nevertheless, some of the results already obtained are so important that I have decided to condense in this paper an outline of many closely-written pages of experimental notes on this subject. I hope to find time hereafter for a complete outline of the details.

In this discussion I propose to confine myself to two main points: How does gold dissolve in cyanide solutions? and How is it precipitated from them?

I.—The Solution of Gold in Cyanide Solutions.

1. *The Solution of Metallic Gold in Potassium Cyanide.*

The solubility of gold in cyanide of potassium solutions has been known for over 50 years. Soon after its discovery by Elkington, Faraday and Bagration it was investigated by Elsner,* who stated that he had found that while zinc would dissolve in a potassium cyanide solution with the evolution of hydrogen, gold and silver dissolved in such a solution only in the presence of oxygen. Although not enunciated by him in this form, the following reaction expressing these facts is usually known as the Elsner reaction:

$$4KCy + 2Au + O + H_2O = 2AuKCy_2 + 2KHO.$$

The necessity for the presence of oxygen when gold dissolves in cyanide solution has been denied by Mr. J. S. MacArthur, who claims the honor of the invention of the cyanide process. In his article on "The MacArthur-Forrest Process of Gold Extraction," read before the Society of Chemical Industry, March

* *J. f. Prakt. Chemie*, xxxvii., p. 441, year 1846.

31, 1890, and reprinted and circulated by the "Gold and Silver Extraction Company of America," Mr. MacArthur says:

"Not having seen the original account of Elsner's researches, I am not in a position to criticize his experiments, but I never could find that the presence of oxygen was necessary either to dissolve gold by itself or from ores by cyanide."

He cites no proof of his opinion regarding the action on metallic gold, but confines himself to speculation, as follows:

"If a piece of gold be immersed in a cyanide solution, so that air, to act on it, would have to penetrate two or three inches of the solution, the gold will dissolve in its usual slow and steady fashion. The equation shows that either oxygen must be absorbed or hydrogen evolved. I have seen no evidence of the former, and can adduce no proof of the latter; but I think the latter more probable, because I cannot conceive oxygen penetrating even a film of cyanide solution without oxidizing the cyanide to cyanate, whereas, in the other case, as suggested to me by my friend, Mr. Ellis, the nascent hydrogen may be at once seized by the excess of cyanide present and ammoniacal compounds formed."

Unfortunately for this ingenious speculation, it does not seem to be borne out by the facts. Maclaurin* has shown by a very able research:

(1) That oxygen is necessary for the solution of gold in potassium cyanide solutions, and that it combines, quantitatively, according to Elsner's reaction.

(2) That the rate of the solubility of gold in potassium cyanide solutions passes through a maximum in passing from concentrated to dilute solutions.

(3) That this remarkable variation is explained by the fact, which he also proves, that the solubility of oxygen in cyanide solutions decreases with the concentration of the latter.

But, to continue with our quotation from Mr. MacArthur, he says:

"However, we do not concern ourselves much with the reactions of pure gold, but as a matter of fact we cannot find that oxygen plays any part in the cyanide-extraction of gold from ores. We have treated an ore with cyanide with free access of air, and then a parallel experiment was done with boiled water, the bottle filled to the stopper with solution and ore, and the stopper sealed. The extraction was the same in both cases."

It will be seen that nothing is said as to the composition of the ore, so that the experiment is not decisive.

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* *Jour. Chem. Soc.*, xliii., 724, May, 1893.

While Maclaurin's experiments seemed conclusive, still the difference of opinion between Elsner and Maclaurin on the one hand, and the reputed discoverer of the cyanide process on the other, was so fundamental that it seemed worthy of further investigation. The importance of this fundamental reaction is not merely scientific, it is of the greatest practical importance in the application of the process.

Hence, this was one of the first points to which I turned my attention. The result of my investigation was an entire confirmation of the accuracy of the Elsner reaction. That is, repeated experiments indicated that a solution of pure cyanide of potassium in pure water, from which all other substances are excluded, is entirely without action on metallic gold. Under favorable circumstances, such a solution absorbs oxygen from the air (without, as MacArthur assumed, immediately oxidizing the cyanide to cyanate), and the affinity of the potassium for oxygen and water, combined with the affinity of the cyanogen for gold and cyanide of potassium, leads to the formation of caustic potash and potassium aurocyanide, as per Elsner's reaction. When the air present is limited, the reaction stops when the oxygen of the air is exhausted, and begins again when it is supplied.

2. *The Effect of Oxidizing Agents.*

This point being clearly proved, the question next arose: Would not other oxidizing agents, capable of liberating nascent cyanogen in a cyanide solution in the presence of gold, also cause its solution? Several hundred experiments showed that this assumption was likewise correct. Nearly all the oxidizing agents were found to be effective. Among those used were: Potassium chlorate, nitrate, permanganate, bichromate, and ferricyanide, and the peroxides of barium, manganese, lead, and sodium. Each of these, and other oxidizing agents, when added to a potassium cyanide solution, increased the solubility of the gold over that shown in a similar solution without such addition. Among the most convenient and efficient of these agents were peroxide of sodium, peroxide of manganese, and ferricyanide of potassium. These experiments were made in 1892–3. It now became apparent that there were many analogies between the cyanide process and the chlorination process. In

the chlorination process the gold requires only free chlorine and water for its solution, as the chloride of gold is soluble in water, but in the cyanide process there are required, first, free cyanogen to form the cyanide of gold, and then free cyanide of potassium to form the soluble double cyanide.

As these facts became clear, other means of producing nascent cyanogen in the presence of metallic gold and a solution of potassium cyanide suggested themselves. Cyanogen bromide was the first of these agents that proved effective, in experiments made January 19, 1894. When cyanogen bromide is added to a solution of potassium cyanide the following reaction ensues:

$$BrCy + KCy = KBr + 2Cy.$$

In the presence of metallic gold and an excess of potassium cyanide, the further reaction ensues:

$$2KCy + 2Au + 2Cy = 2KAuCy_2.$$

Or, as the two reactions are in reality simultaneous, this may be expressed, as has already been done by Mr. Sulman* as follows:

$$3KCy + 2Au + BrCy = 2KAuCy_2 + KBr.$$

The success of the above experiment was so great that ten days later, January 29, 1894, a simpler process was tried. This was, to add a small amount of dilute bromine water to a dilute stock-solution of potassium cyanide, the latter in chemical excess, in the presence of metallic gold.

* Messrs. Sulman and Teed claim (in the *Eng. and Mining Journal*, February 23, March 30 and April 20, 1895) to have made the first discovery of this reagent, and they received an English patent September 14, 1895. Whether their work antedates mine or not, I do not know. A. W. Warwick, in a valuable paper in the *Eng. and Min. Journal*, June 29, 1895, claims to have made the same discovery independently, but does not give the date, and seems to misunderstand the real nature of the reaction that takes place, assuming that oxygen must be present when cyanogen bromide acts in a solution of potassium cyanide in the presence of gold. The *Mining and Scientific Press* of March 2, 1895, says that some months previous to that date a letter had been received from an assayer in northeastern Washington claiming that bromine could be substituted in the cyanide process. E. A. Schneider, *Eng. and Min. Journal*, March 23, 1895, makes a similar claim. It would, therefore, appear doubtful whether Messrs. Sulman and Teed are the original discoverers of the bromo-cyanogen process.

The reactions that take place may be analyzed as follows:

$$(a)\ 2KCy + Br = KBr + Cy + KCy.$$
$$(b)\ Au + Cy + KCy = KAuCy_2.$$

Or, as the reactions under favorable circumstances may be simultaneous:

$$(c)\ 2KCy + Au + Br = KAuCy_2 + KBr.^*$$

Of the two reagents, bromo-cyanogen and dilute bromine-water, I distinctly prefer to add the latter to the potassium cyanide solution, in spite of the opinion of Mr. Teed† to the contrary. The use of the dilute bromine-water is much the simpler method, as the reagent is easily made of any desired strength and may be added as needed, care being taken always to have the potassium cyanide in excess, and not to add the bromine-water faster than the cyanogen set free can be absorbed by the gold. To do this correctly requires experience and skill; and unless the work is in the hands of a competent metallurgist, a great loss of cyanide and poor results will follow, whether bromo-cyanogen or bromine-water alone is added to the stock-solution of potassium cyanide. The reason of this appears to be the great tendency of free cyanogen to form para-cyanogen, to oxidize into certain obscure compounds of a dark color, and to "run down" generally. Hence, if the cyanogen is set free faster than the gold can absorb it (owing to lack of contact at the time that the cyanogen is set free), a loss of cyanogen takes place to no useful purpose. Strong solutions of bromine and cyanide of potassium should be avoided for like reasons.

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* In the *Eng. and Min. Journal*, June 1, 1895, in an article on "A Bromo-Cyanide Process for Gold Extraction," Mr. C. A. Mulholland offers the following provisional explanation of the reaction which takes place when bromine is added to cyanide-solutions in the presence of gold:

$$4Au + 8KCy + 2Br + O_7 + H_2O = 4AuKCy_2 + 2KBrO_3 + 2KOH.$$

This very involved reaction appears to have been imagined because it was supposed that gold could not dissolve unless oxygen was present, the author having lost sight of the fact that the bromine is capable of doing the same work as oxygen.

† *Eng. and Min. Journal*, April 20, 1895.

The ideal requirement seems to be to secure that intimate molecular contact which is assumed in equation (*c*), above. In this case the cyanogen reacts in the nascent state. This is a condition hard to realize in the treatment of ores.

In the treatment of gold strips in cyanide-solutions, I found that with a given cyanide solution and a given bromine-water solution, the amount of gold dissolved by the combined treatment with cyanide and bromine-water was, in one case, four times as great as the sum of the amounts of gold dissolved by the same amounts of cyanide and bromine-water solutions, each acting separately, all the other conditions being the same.

Chlorine-water may be used instead of bromine-water. According to the reaction in equation (*c*), 1 ounce of bromine as such will cause the solution of 2.45 ounces of gold, while 1 ounce of chlorine will cause the solution of 5.52 ounces of gold, and, as per Elsner's reaction, 1 ounce of oxygen will dissolve 24.5 ounces of gold. Weight for weight, the oxygen is more effective than either chlorine or bromine. I have also found that, properly applied, the oxygen of the air gives as good and even better results than either chlorine or bromine. But the latter, being more soluble in water, are more conveniently used.*

It may be concluded from these experiments, that cyanide of potassium solutions, acting alone, are absolutely without action on metallic gold; that the addition of some substance capable of taking the potassium away from a part of the cyanogen, thus setting the latter free in the nascent state, is an essential condition to the solution of metallic gold in potassium cyanide;

* A possible application to the chlorination-process suggests itself. Owing to insufficient roasting or imperfect gassing, the tails from the chlorination-vats often run high. At present these must either be thrown away or else dried, re-roasted and regassed and leached. It might be feasible to have a set of large tailing-vats into which such lots might be thrown wet and leached with a dilute solution of cyanide until they were sufficiently low in gold to throw away. The small amount of chlorine remaining in such tails would act to set free cyanogen enough to fix the gold. Care should, of course, be taken to wash out soluble metallic salts and to neutralize the free acid before adding the cyanide. Such a method is likely to fail in the hands of Mr. Stetefeldt's old enemy, "the muscular amalgamator." And it is doubtful whether sintered clots, formed in roasting, and coarse gold which has escaped the amalgamator, would yield to this treatment.

and that this agent may be either the oxygen dissolved from the air, or supplied artificially, or a suitable oxidizing agent, such as some of those already mentioned, or an equivalent addition of chlorine, bromine, or iodine, or the compounds of these with cyanogen.

Which of these should be used will depend on many conditions, hard to specify briefly. There is danger in using too strong an oxidizer or too much of it. In such a case, not only are other secondary reactions likely to ensue with ores containing metallic sulphides, arsenides and antimonides, which lead to the destruction of the cyanide, but, as already pointed out, the free cyanogen itself, if set free faster than it can be taken up by the gold, simply runs down into oxidized products without doing any good. This is the great difficulty in the treatment of low-grade ores. Nothing but experience and intelligent and trained supervision can meet this difficulty.

One cannot avoid noting a certain analogy between the action of oxygen of the air on the blood and on the cyanide-solution. In the former the oxygen is held by the red corpuscles in a state of readiness for combination, but it is not actually used for combustion except under the nervous stimulus which determines combustion at the point where energy is to be produced. In the cyanide-solution, the oxygen and the cyanide of potassium may exist side by side in solution (as experiment shows) without sensible action on each other, unless the presence of gold determines the Elsner reaction at the point where oxygen, water, gold, and cyanide of potassium meet. If a thorough utilization of the cyanide is desired, the action of the oxidizer should be such that the cyanide will be decomposed only under the stimulus of the presence of the gold. Under these circumstances, the cyanogen will be utilized to the utmost. If, however, the oxidizer is a powerful one, capable of attacking the cyanide on its own account in the absence of gold, the inevitable consequence will be a rapid destruction of the cyanide, just as would be the destructive effect of such oxidizers on the blood.

It cannot be too much emphasized that it seems to be nascent cyanogen only that acts on the gold. I have tried saturating a cyanide of potassium solution with free cyanogen, but have usually obtained only very unsatisfactory results. The ten-

dency to form paracyanogen probably explains why the cyanogen in this case has so little effect on the gold. The further tendency of the paracyanogen to absorb oxygen and run down explains also why its presence in quantity is not only of no use but is a positive detriment, as it prevents the oxygen from setting free nascent cyanogen which can attack the gold.

On the whole, with low-grade ores and dilute solutions, the cyanide-solution itself will, if properly aerated, carry oxygen enough to dissolve the gold; so that artificial oxidizers are seldom needed, unless there is some reducing agent present in the water or the ore, which absorbs the oxygen. In such a case, and with richer ores and stronger solutions, there is sometimes a distinct advantage in the use of oxidizing agents. But, unless used with the nicest discrimination, they do more harm than good.

3. *The Solubility of Gold Sulphide in Potassium Cyanide.*

There still remains a great difference of opinion as to the condition in which gold occurs in its ores. There is no doubt that the greater part of it occurs in the metallic state. Many contend that it is always metallic. It cannot be denied, however, that it occurs combined with tellurium, and there is some evidence that it occurs combined with other sulphides, antimonides, arsenides, and, perhaps, with bismuth, to form compounds little understood, but of great practical importance in the treatment of sulphide ores containing gold.

It seems to be impossible to combine gold with sulphur alone in the dry way. I have, therefore, tried the sulphide of gold, Au_2S_2, thrown down from the acid chloride by sulphydric acid gas. In a 5 per cent. solution of potassium cyanide this sulphide dissolved completely in four minutes; in a 1 per cent. solution, in ten minutes; in a one-fifth per cent. solution, in one hour. In all cases there were found in solution sulphide of potassium (K_2S) and sulphocyanate of potassium (KCyS), and, in most cases, suspended sulphur. The latter was most apparent with the dilute solutions of cyanide.

The reactions which occur seem to be expressed most simply by the following formulas:

$$Au_2S_2 + 4KCy = 2AuKCy_2 + K_2S + S.$$
$$KCy + S = KCyS.$$

A third reaction seems indicated by some of the quantitative results, viz.:

$$Au_2S_2 + K_2S = Au_2K_2S_2 + S.$$

The exact relation of these three reactions seems to depend on the strength of the cyanide-solution used. The remarkable solubility of the sulphide of gold is no doubt partly due to the greatly increased surface of the sulphide exposed to the attack of the solvent, but it is also due, no doubt, to the double affinities set up between the gold and the cyanogen on the one hand, and the potassium and the sulphur on the other. There is evidently no call here for oxygen or an oxidizing agent to increase the solubility of the gold.

4. *The Solubility of Gold Telluride in Potassium Cyanide.*

It being very difficult to prepare the telluride of gold in a wet way similar to that in which the sulphide of gold was produced, I have experimented with telluride of gold prepared by fusing pure gold with tellurium, as well as with natural minerals containing tellurium and gold and silver.*

The result of the experiments so far made has been to show that the tellurides of gold and silver experimented on both dissolve in the cyanide solution with the greatest difficulty. This is probably due to the fact that the affinity of potassium for tellurium is much smaller than for sulphur; but tellurium, tending to absorb oxygen and forming tellurous acid, also tends to retard the oxidation of the potassium and the solution of the gold.

5. *The Solubility of Antimonide of Gold in Potassium Cyanide.*

An antimonide of gold produced by fusion of the pure metals acted very much as did the telluride of gold. A copious white powder of antimony oxide formed in the solution and the action was retarded, probably for the same reasons as operated with the telluride of gold.

* I have had great difficulty in obtaining material of this kind of sufficient quantity and purity for the work, and would be greatly obliged to members of the Institute who will supply me with adequate material. What I particularly desire is the pure tellurides of gold and silver existing separately in such quantity that it will pay for the outlay of time and labor necessary for analyses and experiments.

II.—The Precipitation of Gold from Cyanide Solutions.

Those who are familiar with the instability of most of the salts of gold, and the ease with which all reducing agents, and even light, precipitate metallic gold, will be surprised at the tenacity with which gold is held in solution as aurocyanide of potassium. None of the usual agents can be relied on to precipitate it; sulphydric acid, oxalic acid, sulphurous acid, ferrous sulphate, etc., which are usually so prompt in precipitating gold, are of no avail here. So long as the cyanide of potassium is in excess, they give either not the least trace of a precipitate or only a very imperfect one. In order to have these agents act with certainty, it is necessary to destroy all the cyanogen present. This is a long and difficult process. My attention was early directed to the need of a certain method of recovery, for precipitation by zinc-shavings was exceedingly inconvenient in laboratory tests. For a long time the only certain method that could be relied upon was either to evaporate the whole solution to dryness in a dish made of test-lead, and to scorify the residue, or else to acidify the solution with sulphuric acid, and boil till copious fumes of sulphuric acid appeared and the gold separated out in the metallic state. The solution was then diluted, filtered and the residue scorified. The results were satisfactory, provided the evaporation was pushed far enough to destroy all the cyanide, and no chlorides were left in the solution. Otherwise the results were liable to be low. The extreme tedium of this method, even on a laboratory scale, led very early in the study of the solubility of gold to a parallel study of the causes that contributed to the opposite effect, viz., the precipitation of gold from its solutions. In reality, these are the two opposite aspects of the same question.

According to most of the authorities, when a solution of potassium aurocyanide is acidified with sulphuric acid the cyanide of gold promptly separates as a yellow precipitate. From the solution obtained in treating gold-ore with a 1 per cent. cyanide-solution, even when it contains several hundred dollars' worth of gold per ton, nothing of the sort takes place. The solution remains perfectly clear and colorless. It is only when the solution has been evaporated down and the acid gets concentrated and hot that the canary-yellow aurocyanide separates out. If the acid gets stronger, this salt is gradually decom-

posed, and finally metallic gold remains. The cyanide of gold is more difficult to decompose than the telluride. For it is only the strongest hot sulphuric acid which will convert the canary-yellow crystals of AuCy into metallic gold.

The remarkable fact that the gold remains in solution after being acidified with a strong mineral acid, and that a precipitate forms only on the long-continued application of heat, seems to show that a compound $HAuCy_2$, which I shall call aurocyanhydric acid, is formed, similar to the ferrocyanhydric and the ferricyanhydric acids; in short, that we have to deal with an acid radical of the same class as the two just named, and not with an ordinary double cyanide, AuCy, KCy, as is usually supposed. There are many converging indications pointing to this conclusion which it would take too long to enter into here.

1. *Regeneration of Potassium-Cyanide from Dilute Solution by Acidification.*

Before this fact was understood, it was thought that the gold was kept in solution by the free HCy left in the solution after acidification by sulphuric acid. If this free acid could be removed, it was hoped that the gold cyanide would be precipitated. At the same time it was hoped that the free HCy could be regenerated by absorption in caustic potash. This subject was exhaustibly investigated in February to May, 1893.

It was found that although, when acidified with sulphuric acid, a 1 per cent. solution of potassic-cyanide gave not the least apparent trace of escape of HCy, yet, if the solution thus acidified was left for several weeks near an open vessel containing caustic potash, nearly all the HCy would be absorbed by the alkali. As this operation was too slow to be of practical use, air was pumped first through the acidified solution and then through the solution of caustic. The same air was used over again to avoid the presence of carbonic acid. By using steam with the air, the contents of the stock-solution in cyanide were reduced in three hours from 1 per cent. to 0.07 per cent. of free HCy. By using absorption-towers (the usual means of increasing surface-exposure) such results might be reached in practice without the use of steam. Similar results were reached, without the use either of absorption-towers or of steam, in from eight to ten hours.

At the same time it was found possible to regenerate from a 1 per cent. stock-solution of cyanide, a concentrated solution of any desired strength of potassium- or calcium-cyanide, with a total of cyanide actually recovered, up to 80 or 90 per cent. The avidity with which caustic potash or lime absorbs the HCy reduces the loss to a minimum. In case the solution contains alkaline sulphides, these would have to be removed previously by treatment with lead carbonate or sulphate.

A method was thus found which, with a little engineering skill, can be used for regenerating solutions of cyanide of potassium.

2. *The Precipitation of Gold from Aurocyanhydric Acid.*

Although it was thus found possible to remove, down to 0.01 per cent., the free HCy from such a solution acidified with sulphuric acid, still the gold, even to the amount of $603 per ton, or 0.1 per cent., would remain in the perfectly clear, colorless solution. As already stated, the only explanation found for this was the presence of aurocyanhydric acid ($HAuCy_2$) in solution. It was found that, even under these circumstances (absence of all free KCy, and only a trace of free HCy), all the usual reagents, such as oxalic, sulphurous, sulphydric acids, etc., failed to precipitate the gold, either giving no precipitate or else only a very imperfect one.

Experiments were made as early as March, 1893, with nitrate of silver, nitrate of lead, and mercurous nitrate. A complete precipitation of the gold resulted when an excess of nitrate of silver was added, either to the above aurocyanhydric acid or to the corresponding potash salt. This would be a neat and practicable method on the large scale, if it were not for the cost of the silver. The nitrates of lead and mercury gave partial precipitation in most cases.

Oxidizing agents, such as permanganate of potassium and peroxides of manganese and lead in acid solutions, also facilitate the precipitation. The filtration of the solution through red lead or massicot has been covered by an English patent granted to P. de Wilde, December 22, 1894.

3. *Precipitation of Gold from Potassium-Aurocyanide by Means of Charcoal.*

On July 3, 1894, an American patent was granted to the late

William D. Johnson, at that time chemist of the California State Mining Bureau, for a method of extracting gold from a cyanide-solution by passing the solution through a series of charcoal filters, and subsequently burning the charcoal and smelting the ashes with suitable fluxes.

The use of charcoal for this purpose had already been tried for extracting gold from the chloride solutions produced in the Plattner process. For this purpose it is certainly a very efficient method of precipitating gold; but the subsequent extraction of the gold from the charcoal is almost as troublesome as the extraction of the gold from the original ore.

Dr. Johnson claimed that a single filtration would remove 25 per cent. of the gold, and that, by repeated filtrations, 95 per cent. of the gold could be recovered. It is evident that if only 25 per cent. of the gold present could be recovered by a single filter, it would take a very large number of filters to remove all the gold, and consequently a large volume of charcoal would have to be burned to recover it.

In order to test the method, the following experiments were tried: Two cyanide-solutions containing each 100 c.c. with about 0.1 per cent. of gold ($603 per ton solution), were each filtered through separate filters, and the process was repeated four times. Each filter contained seven grammes of charcoal. One of the solutions was alkaline, the other had been made acid with sulphuric acid. From the acid one 15 per cent., in the other case 6.5 per cent., of the gold present was precipitated by the charcoal. These experiments, made under much more favorable conditions than could occur on the large scale, show that this method can hardly prove of service with rich solutions in practical work.*

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* Since this paper was presented at the Denver meeting, I have been paying more attention to the action of dilute cyanide solutions, and I have thought it well to investigate the action of charcoal on such solutions.

For this purpose the following experiments were outlined: Two solutions of 2000 c.cms. each were prepared. The one (*a*) contained 0.001 per cent. of gold as potassium aurocyanide, or $6.03 per ton solution, but no free cyanide of potassium. The other (*b*) contained the same amount of gold, but also 0.2 per cent. of cyanide of potassium. Each solution was contained in a Mariotte bottle 11.4 cms. in diameter, so arranged that the solutions were discharged upon charcoal filters so that the charcoal was kept constantly submerged, and under a constant head. The charcoal was crushed so that it passed a 20-mesh sieve, but was mostly

4. *Precipitation of Potassium-Aurous Cyanides by Mercuric Chloride.*

Fremy (*Encyclopédie Chimique*, t. 2, *Métalloïdes*, Sec. 2, Fasc. 2, p. 463) states that potassium-aurocyanide is precipitated by mercuric chloride according to the following reaction:

$$2KAuCy_2 + HgCl_2 = 2AuCy + 2KCl,HgCy_2.$$

This reaction I have also tested. A solution assaying several hundred dollars per ton, and carrying only a trace of free cyan-

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fine dust. It was contained in a cylindrical glass tube, 10 cms. long by 2 cms. in diameter; it weighed in each case dry 10 grammes.

It took nearly two days for the first 800 c.cms. of solution to filter through each filter. The filtrate from (*a*) contained no gold. That from (*b*) contained a trace (less than 0.01 mg.). The next 1200 c.cms. took nearly five days to filter. Filtrate (*a*) contained 0.01 mg. of gold, while (*b*) contained 1.40 mgs. The charcoal from (*a*) contained 20.03 mgs. gold, while that from (*b*) contained 18.70 mgs.

The actual recovery of the gold by the charcoal was in (*a*), in the absence of free cyanide, 99.95 per cent.; in (*b*), in the presence of 0.2 per cent. free cyanide, it was 93.03 per cent. The free cyanide, in the latter case, was reduced from 0.2 per cent. to 0.118 per cent.

These results are of great importance, for they show that, under certain conditions, all the gold can be precipitated from potassium aurocyanide by means of charcoal, and that even in the presence of free cyanide of potassium 93.03 per cent. can be precipitated.

Further study of the data given above shows that the charcoal in (*a*) contained 0.2 per cent. gold, or $1206 per ton, while (*b*) contained 0.187 per cent., or $1182 per ton. These figures would seem to represent the limit of enrichment at which the charcoal ceased to act in either case, the limit being lower in the presence of the free cyanide. In the case of the stronger solutions previously tested (which were 100 times richer in gold), the content of the charcoal in the acidified test was 0.21 per cent. gold, or $1266 per ton; and in the alkaline one, containing a certain amount of free cyanide, it was 0.091 per cent., or $548 per ton.

These results would seem to show that Dr. Johnson's view that the charcoal can be depended on to precipitate 25 per cent. of the gold present at each filtration is erroneous. It would rather seem that a given amount of a certain charcoal has a definite capacity of precipitation; that within this limit it acts completely and promptly; beyond this limit it acts less completely and quickly; and that finally it ceases to act at all. The presence of free cyanide of potassium seems to lower its precipitating capacity, and perhaps at a certain point stops it altogether; acidification seems to increase the capacity. These views are, of course, provisional, as the subject is still under investigation.

It must be evident that in its present form the method gives a much less concentrated precipitate of gold (only 0.2 per cent.) than either the zinc, the electric or the cuprous method of precipitation. The precipitate obtained in the latter method often contains 40 to 60 per cent. of its weight in gold, or it assays from $240,000 to $360,000 per ton.

ide of potassium, remained perfectly clear for several hours after the addition of mercuric chloride. The next day, however, I found a yellow precipitate of cyanide of gold, so fine that it filtered clear with great difficulty. When, finally, a clear filtrate was obtained, the gold continued to precipitate for several days; it also had a tendency to adhere strongly to the walls of the tube. These indications seemed to point to the need of heat to hasten the reaction. Consequently I tried heating the solution. I found that this greatly aided the reaction. On boiling the solution for a few minutes the gold was almost entirely thrown down as the yellow cyanide, and could be easily filtered out, leaving a perfectly clear filtrate.

In the filtrate from the aurous cyanide the mercury could be saved and the cyanide recovered by adding, very exactly, the amount of sulphide of potassium to throw down the mercury as sulphide. Cyanide of potassium would thus be regenerated in the solution. The latter would then be ready for another leaching; and even should a little gold be left in the solution from imperfect precipitation it would not be lost. The precipitated cyanide of gold, gently ignited at a low red heat in the air, turns to pure gold.

This method of precipitating may be useful under certain circumstances, but the necessity of heating the solution adds so many complications and so much expense to a leaching-process, that it is robbed of many of its advantages. In the presence of free cyanide of potassium this precipitation does not take place.

5. *On the Precipitation of Potassium Aurocyanides by Copper and its Compounds.*

It was early noted in the study of the cyanide-process that copper, with its compounds, was destined to play a very important rôle in the development of the process. In fact, it is well known that certain copper-minerals are active obstacles to the successful extraction of gold from its ores.

The thought naturally suggested itself: May not this difficulty in the solution of gold be utilized to aid in its precipitation? In March, 1893, a sheet of metallic copper was immersed in a solution of cyanide of potassium containing a third of a gramme of gold in solution. Instead of the precipitation of metallic gold on the copper, a light yellow precipitate began to

form throughout the liquid. Very little of this formed on the copper itself; it was found as a sediment throughout the liquid. After three days three-fourths of the gold had been precipitated. The copper was then removed; but the gold-containing sediment continued to separate from the solution for over a week, until all but 11 mgs. of the gold had been precipitated. This very important experiment did not at the time seem to lead to anything practicable, for the action seemed too slow and uncertain to be of any value.

Recalling the fact that the sulphide of copper had been suggested by C. H. Aaron and L. D. Godshall (*Eng. and Min. J.*, Nov. 29, 1890) for the precipitation of gold from the chloride-solution produced in the Plattner chlorination-process, it occurred to me that a similar precipitation might take place from the cyanide-solution. Hence, on May 4, 1893, three lots of potassium aurocyanide, with 0.02 per cent. of KCy and a quarter of a gramme of gold, were treated by agitation for four hours with CuS precipitated from $CuSO_4$ by H_2S. The still moist precipitate was washed and then stirred into the solution. One of the solutions was left neutral, the other made strongly alkaline with caustic potash, and the third made acid with sulphuric acid. In the alkaline and neutral solutions two-thirds of the gold were thrown down by the copper sulphide; in the acid one, all of it was precipitated. Numerous experiments with the sulphide of copper produced by fusion showed that the form originally used gave the best results, probably on account of its greater surface. Since the date of its first discovery, in May, 1893, this method of recovery of gold from cyanide-solutions has been repeatedly used in my laboratory, and has stood the most rigid tests. It is necessary to acidify the solution with a mineral acid like sulphuric acid, and it is best to remove as far as possible the free HCy by one of the methods already explained. In case there is less than 0.1 per cent. of free HCy present this may be omitted. The CuS,Aq produced by the action of H_2S on a solution of $CuSO_4$ is then added to the solution; the whole is stirred thoroughly at intervals for several hours and then filtered out. The gold will be contained in the residue.

In one case, CuS, weighing, when dried, 5 grammes, was added to 1000 c.c. of an acidified solution of aurocyanhydric

acid containing 1 gramme of gold in solution. All the gold was precipitated. The copper sulphide precipitated one-fifth of its dry weight of gold. The gold was readily recovered from the residue, after making it alkaline by digesting it with sodium sulphide, in which the gold is readily soluble. The copper sulphide may then be used over again. From the sulphide-solution the gold is readily precipitated by electrolysis, the more readily as the solution is small in bulk and concentrated. Other methods of extracting the gold from the copper are also available.

The gold may also be removed from the cyanide-solution, after acidification, by filtering it through CuS without agitation. I prefer, however, to agitate first and filter afterwards. Another method that was used was to add copper sulphate and then sulphydric acid to the acidified solution. Provided the excess of free HCy be first removed, one atom of copper will thus precipitate one atom of gold, or one pound of metallic copper as sulphate will precipitate 3.11 pounds of gold.

Both of these methods were used with complete success on the solutions obtained by treating gold-ores with cyanide solution at various intervals from May, 1893, to 1896.

A careful study of the solutions used in the above experiments showed that the gold was precipitated without more than a trace of copper going into the solution. The unusual occurrence of one metal precipitating another without itself going into the solution, led to a careful study of the reaction involved. This was also viewed in the light of the experiment with the strip of metallic copper already mentioned. Also, in an experiment made in 1894, where Cu_2S was used to precipitate the gold from an acid solution of aurocyanide, a white precipitate, containing cuprous cyanide, was formed, which gave to the entire solution the appearance of milk. It was also noticed that when copper sulphate was added to a strong solution of aurocyanide, a white precipitate formed which contained gold and copper. Also, when H_2S was added to a gold-cyanide solution containing copper, it was noticed that, at the first instant when the gas came in contact with the solution, a white precipitate was formed, which gradually changed to the black color of copper sulphide as more H_2S was added. All these considerations pointed to the conclusion that the gold-containing

precipitate was in all cases mainly a salt of aurocyanhydric acid, either cuprous aurocyanide or the corresponding sulpho-salt.

This explanation of the reaction was made early in 1894. As soon as this was settled, another method suggested itself, and proved to be a success. A solution of cuprous chloride was prepared by adding sodium chloride to a solution of cupric sulphate, and saturating the mixture with sulphurous acid. When this was added to the potassium aurocyanide, a white precipitate was formed at once, which contained all the gold. In the first experiment tried the metallic copper, present as cuprous chloride, precipitated over twice its weight of gold. All the cuprous salts were found effective for this purpose. Even the cuprous cyanide formed by adding copper sulphate to a potassium cyanide solution, and then acidifying with sulphuric acid and filtering out the white, curdy cuprous cyanide, was thus effective. As this salt is much more permanent than the cuprous chloride, it may prove of service in practice, though it is more expensive than the latter. I have also found that cuprous hyposulphite precipitates aurocyanides. The precipitate, however, is soluble in excess of either potassium cyanide or sodium hyposulphite. Hence a complete precipitation of the gold can only be expected after the solution has been made acid.

The method that I have used most constantly since April, 1895, is to add sulphuric acid to gold-cyanide solution till it reacts acid to litmus. If the solution is strong in HCy, it is usually best to remove this first, either by the aeration method already described, or else to remove it, previous to the acidification, by means of the zinc-sulphate method, to be mentioned later. To the acidified solution I add cuprous chloride, as above described. When this solution is added, the gold and excess of HCy come down at once as a white precipitate, generally turning slightly yellow. The cuprous chloride should be added till a drop of the filtrate gives a red precipitate with ferrocyanide of potassium. The gold is thus rendered insoluble, and may be filtered out as cuprous aurocyanide ($CuAuCy_2$). This compound is insoluble in dilute acids, but readily soluble in cyanide of potassium. The gold is easily extracted from it in several ways.

In October, 1895, Prof. P. de Wilde, of the University of Brussels, published a method of precipitating gold from cyanide solutions in the *Revue Universelle des Mines, de la Métallurgie, etc.*, in an article entitled "Note on a New Method for the Extraction of Gold from Tailings, Slimes and Concentrates."

His method consists in three steps:

a. The leaching of the ore, etc. This he does, as in the Siemens and Halske process, with an 0.05 per cent. KCy solution, containing about the same amount of free caustic soda or lime, as may be necessary.

b. The recovery of the excess of alkaline cyanide. This he attempts, as had already been proposed by MacArthur, by adding ferrous sulphate and precipitating ferrous cyanide until potassium ferricyanide gives with a drop of the filtrate a precipitate of Prussian blue. In order to avoid the precipitation of the gold it is necessary that the free alkali should be nearly neutralized before adding the ferrous sulphate. This method will work, as I have proved by experiment, but I regard it as thoroughly unpractical. The bulk of the precipitate is something enormous; it takes a week to filter and remove the gold-solution; it is constantly changing composition, running finally into Prussian blue; and it is no simple matter to extract the cyanide from it when once it has been obtained. The author himself evidently does not put much faith in it. He concludes by remarking that with dilute cyanide-solutions this step of recovery may be omitted.

c. The precipitation of the gold. Here he describes a method the same in principle, and almost the same in detail, as the one I have described as used in my laboratory since April, 1895. I translate from his article:

"My third operation is based on the following principle: In a solution containing the double cyanide of gold and potassium ($AuKCy_2$) (or sodium) one precipitates all the cyanogen in the form of aurous cyanide (AuCy) and cuprous cyanide (CuCy), on acidifying first the solution with sulphurous acid (SO_3H_2) and then adding a solution of sulphate of copper ($CuSO_4$).

"The reaction is extremely sharp, and the least excess of sulphate of copper suffices to produce an absolutely complete precipitation of the gold. In operating under the most varied conditions and with solutions containing only five or six milligrammes of gold to the 1000 c.c., I have never been able to find the least loss of gold.

"If the liquid contains alkaline cyanide in excess (which is the case in my method), this is transformed completely into cuprous cyanide.

"The acidification can be made by the injection of sulphurous anhydride (SO_2) by means of a solution of sulphurous acid, or of an alkaline bi-sulphite (meta-sulphite of potassium or of sodium $Na_2S_2O_5$)."

It will be seen that Prof. de Wilde makes his cuprous salt in the solution itself, also acidifying the solution at the same time with sulphurous instead of sulphuric acid. This may appear an advantage; I think, however, that the mechanical difficulty of treating a large volume of solution with gas will more than offset the slightly greater cost of the sulphuric acid for the same purpose. However this may be, I prefer my method of making the cuprous salt outside of the stock-solution and from strong solutions, so that the application of the sulphurous acid may be more readily controlled and the production of the cuprous salt more complete. With these slight differences of detail, the methods are the same. Whether the method was first discovered in Belgium or California is a mere matter of dates.

I have tried the method suggested by Prof. de Wilde, as well as my own, with solutions containing from only \$3 to over \$600 per ton of gold (from $\frac{1}{2000}$ of 1 per cent. gold up to $\frac{1}{10}$ per cent. gold), and I have found them both to remove the gold with a sharpness and completeness that is a great relief, after trying the other methods in use.

I find, however, that with Prof. de Wilde's method it is necessary to let the solution stand at least twelve hours before filtering, otherwise all the gold may not be precipitated. In fact, in some cases not the slightest trace of a precipitate will form for several hours, the liquor remaining clear as crystal. At last, however, it never fails to come down. Gently heating the solution hastens the reaction; cold retards it. At the ordinary temperature of the air (60° F.) heating is not necessary for the completion of the reaction, but more time must be then allowed. It is best, however, to allow the filtrate to stand for several hours, another twelve hours if possible, to be sure that no further precipitate comes down. This is where my method has the advantage; being formed in concentrated solutions, my cuprous salt is already reduced and the precipitate forms sooner. I think the reason for the delay in his method is the greater dilution of the copper salt and the sulphurous acid, which retards the formation of the cuprous salt.

I feel sure that this method in either of its forms will prove a valuable addition to our present means ot recovering gold from cyanide-solutions.

But before the advantages of this method can be duly appreciated, it will be necessary to briefly consider the methods for precipitating gold at present in use. There are only two which have made any headway on a working-scale, namely, the use of zinc-shavings, as recommended by the MacArthur-Forrest people, and the use of electricity, as in the Siemen and Halske process. I shall consider them in inverse order.

6. *The Use of Electricity for Precipitating Gold from Cyanide Solutions.*

The use of electricity for depositing gold and silver for electroplating has been practiced successfully for so many years that it would seem that this method would also be a simple, expeditious and practical method of precipitating gold from the cyanide-solutions obtained by leaching gold-ores. But a little reflection will make it clear why so many processes for this purpose which have been patented have never been heard of since.

In electroplating, it is possible and practicable to use: 1, soluble gold anodes, by which the counter-electromotive force of the precipitated metal is entirely neutralized; 2, concentrated solutions of the double cyanide of gold and potassium, so that the specific resistance of the solution is low; 3, an electrolyte of constant composition, whereby the proper working-conditions may be constantly maintained after they have once been reached.

None of these conditions can be secured in precipitating the gold from a cyanide ore-extraction solution.

It is true that in the Siemens and Halske method a soluble anode of iron plate is used; but all the cyanogen that combines with the iron is lost, ferrous cyanide and finally Prussian-blue being formed to no useful purpose. The principal difficulty, however, is one that cannot be obviated, viz., the high specific resistance of the dilute solutions that must be used. With solutions always containing less than 1 per cent., and sometimes less than 0.05 per cent., of free cyanide, and perhaps 0.001 per cent. of gold, or less, the resistances are something enormous, and with any other metal than gold no one would think of pre-

cipitating such a solution with electricity. The high specific resistance of the solution may be met by using an increased surface of anode and cathode and by artificial circulation, but this can only be done at an outlay and with a complication which robs the method of its cheapness, simplicity and convenience.* It is true that electrolytic methods of precipitation are used in the analytical laboratory as convenient methods of analysis. But a long experience with these methods on the small scale has convinced me of the great difficulties which will always stand in the way of their application on the large scale. It is one thing to electroplate an article with gold in a strong solution of aurocyanide of potassium with a gold anode which constantly adds as much gold to a solution as is precipitated, and it is quite another to precipitate all the gold from a solution with an insoluble, or even an iron, anode when the solution contains perhaps 0.05 per cent. of cyanide of potassium and perhaps 0.001 per cent. of gold. Here the problem is to reduce the gold-content to 0.00001 per cent. (or from $6 to 6 cents per ton solution). Any one who has attempted this recovery (which is 99 per cent. of the gold contained in solution), without at the same time destroying most of the cyanide present, will agree with me that it is no easy task.

There is another difficulty that I have met with. Under certain circumstances the gold is precipitated not only on the cathode, but also on the anode; and, in addition, in many cases of the treatment of the solutions produced from ores a precipitate settles out from the electrolyte which also contains gold. The reason for the first difficulty seems to be that when the current decomposes the salt $KAuCy_2$, it decomposes it not into $K,Au + 2Cy$, but the K alone goes to the cathode, while the acid radical, $AuCy_2$, goes to the anode. The deposition of gold which appears on the cathode is due to the metallic potassium attacking and reducing gold as a secondary reaction from the solution there adjacent. Great density of current and long treatment will finally throw out all the gold on the cathode, but

* Since writing the above my attention has been called to a corroboration of these views in an article entitled: "Zinc *vs.* Electricity," by Mr. John Yates, *South African Mining Journal*, September 26 and October 3, 1896. In this article an interesting comparison of these two methods of precipitation is given.

there is enough force in this tendency to greatly retard the electro-deposition of the gold.*

I have been able, by acidifying the solution with dilute sulphuric acid, to get fairly complete precipitation in a reasonable time, say 12 to 24 hours, on a small scale, both with insoluble anodes (carbon, platinum or lead) and with soluble ones (zinc, iron or copper). Of course, in this case the cyanide is destroyed, unless special steps are taken to recover it. The precipitation is much more rapid and complete from these dilute gold-solutions when the reaction is acid than when it is alkaline or neutral.

The difficulties which I have pointed out in the electrolytic method may be met by great technical skill, as has been the case in the Siemens and Halske process, and in that way, when they are reduced to a minimum, the method may be crowned with success. But in the absence of such skill it is bound to be a failure, and the difficulties are of such a serious and fundamental nature as to lead to the desire for another and simpler plan.

7. *The Precipitation of Gold from Cyanide Solutions by Means of Metallic Zinc.*

The precipitation of gold from cyanide-solutions by means of zinc-shavings has been more generally used on the large scale than any other. It appears on the face of it the simplest method that could be devised, and in proper hands and under intelligent supervision it has given, on the whole, better results than any other method in general use. Nevertheless, those who have used it most are the most anxious to find some other method.

Some of the principal objections urged against it are the following:

* A similar state of affairs has been shown to be the case by Professor Hittorf. (Ostwald's *Chemische Energie*, 2te Auf., ii., 880) in the electro-deposition of potassium-silver cyanide. I have also noticed a similar state of affairs with copper. When a solution of potassium cuprous cyanide is electrolyzed with a platinum anode at a voltage below that at which metallic copper is deposited on the cathode, yellowish-brown crusts of cupric cyanide separate out on the anode. These dissolve in the electrolyte when stirred. Fragments of the crusts that drop from the anode on the cathode are reduced to metallic copper. At a higher voltage metallic copper comes down on the cathode from the solution generally. I have also noticed a similar thing in electrolyzing potassium aurocyanide with platinum electrodes. In order to see this action clearly it is necessary to avoid an excess of free KCy.

1. The great consumption of zinc compared with the amount of gold precipitated.

2. The great destruction of potassium cyanide to no useful purpose.

3. The great difficulty of removing zinc and cyanogen residues from the gold, thus causing loss in melting and the production of an unclean bullion.

4. The failure, in certain cases, to precipitate the gold.

I do not need to dwell on the first three of these difficulties. They have been sufficiently emphasized by every one who has described the practical application of the cyanide process; but the incomplete precipitation of the gold has not been sufficiently recognized.

One reason for this appears to be that the reaction which takes place when gold is precipitated from the potassium-aurocyanide is not as simple as it is generally supposed to be. Thus Dr. Scheidell, in the *Bulletin* of the California State Mining Bureau, on the "Cyanide Process," 1894, p. 34, states the usually accepted view. He says:

"The action of zinc on gold-solution is theoretically very simple, a simple substitution of the gold by the zinc according to the equation:

$$2AuKCy_2 + Zn = K_2ZnCy_4 + 2Au."$$

This reaction had also been given in almost the same words by Butters and Clennell (*Eng. and Min. Journal*, Oct. 29, 1892). Wilson ("Cyanide Process," p. 34) and Rose ("Metallurgy of Gold," p. 323) adopt the same reaction.

My attention was first called to this subject by a remark of Mr. W. R. Feldtmann (*Eng. and Min. Journal*, August 11, 1894), who, speaking of the precipitation of gold by zinc, says:

"Its completeness appears to depend, in a measure, on a slight excess of cyanide of potassium being present in the solution."

The same fact was recognized by one of my students, Mr. B. E. Janes, when acting as assayer at the Mercur Mine, Utah (*Mining and Scientific Press*, May 23, 1896).

This led me to investigate the reaction which occurs. I had prepared a solution of potassium aurocyanide with about 1.3 per cent. metallic gold and no free cyanide. A solution made from this by dilution, so as to contain 0.1 per cent., or $603

per ton, gold, was then treated with strips of sheet zinc for twenty-four hours. The strips were carefully burnished with emery-paper, or with a sharp knife, to avoid any film being left on the surface of the zinc. Commercial sheet-zinc was used as is done in practice. The solution was contained in glass tubes sometimes at rest, sometimes rotated mechanically for the whole period of twenty-four hours.

In no case was more than a trace of gold precipitated. In many cases, where the zinc was brightly burnished, not a trace of gold came down, and the strips weighed exactly the same to a hundredth of a milligramme before and after the test, and when dissolved in sulphuric acid left no trace of gold.

These experiments throw light on the statements often made that it has been possible to extract the gold from the ore but not from the solution. A person trusting to the above-described reaction, might easily throw away such a solution after twenty-four hours' contact with bright metallic zinc, even though it contained $603 per ton.

The next experiments were made with lathe-turnings from the same sheet-zinc which, in the polished state, had failed to precipitate gold. With potassium-aurocyanide containing 0.1 per cent., or $603, gold per ton, which had failed to precipitate at all on the bright sheet-zinc, the gold was entirely precipitated from the solution in twenty-four hours by filtering it through a very large excess of zinc-shavings (over four hundred times the weight of the gold present). The filtrations were repeated through the same shavings four times.

When the solution contained only $10-worth of gold per ton, the precipitation was only 82 per cent. of the gold in solution in a single filtration which lasted twenty-four hours. In each case the solution was allowed to remain in contact with zinc-shavings, four hundred times the weight of the gold in solution, until a color of gold appeared on the shavings; then the solution was allowed to flow from the filter drop by drop, the shavings being continuously submerged. The conditions were, therefore, much more favorable for complete precipitation than could be maintained in practice.

These differing results are very difficult to explain on the simple substitution-reaction usually accepted. If the gold precipitates on the shavings, why should it not do so on the same

sheet of burnished zinc from which the shavings were made? There is a larger surface of contact, it may be suggested. But even the smallest fragment of a zinc-shaving will precipitate gold on itself, while, in the same solution the freshly-scraped surface of sheet-zinc, many times its area, is either entirely or practically without action. It may be thought that the turnings had a cleaner surface than the scraped zinc, but this cause was carefully eliminated.

All the indications point to the phenomena of "polarization," so well known in electrolytic work, where the formation of invisible traces of films on the surface of the electrodes puts a stop to further action, either by preventing contact, or by setting up an opposite electromotive force.

I have not had time to establish firmly the following explanation of the phenomena which I have observed, but there seems some evidence in its favor.

There is no doubt that, besides the tendency of the potassium-aurocyanide to split up into KCy and Au and Cy, which is assumed in the reaction by substitution, there is also a tendency for it to split along another line, viz., into K and $AuCy_2$, the latter playing the part of an acid-radical. Now, if this takes place in the presence of zinc and water, the water will be attacked by the potassium, forming caustic potash and hydrogen, and the $AuCy_2$ will be attacked by the zinc, forming zinc-cyanide and metallic gold, according to the following reaction;

$$2KAuCy_2 + 2Zn + 2H_2O = 2ZnCy_2 + 2KHO + 2H + 2Au.$$

There is no doubt that a certain tendency to form this reaction exists, for electrolysis proves it. It can only take place here by setting up a condition of polarization on the surface of the zinc, owing to the film of occluded hydrogen set free on its surface. If this be so, it is now very clear why the zinc turnings act, while the smooth zinc does not. The turnings have an infinite number of ragged edges, which favor the escape of the hydrogen gas, and the relief of the condition of polarization, and thus allow the reaction to proceed. The hard, smooth surface of the rolled sheet-zinc is very unfavorable to the formation of gas-bubbles from its surface, and hence to the continuation of the reaction.

Of course, when the reaction once sets in as above, there is a further reaction between the caustic potash and the zinc cyanide, by which a part of the latter is dissolved, and potassium zinc cyanide and potassium zincate formed. Thus:

$$2ZnCy_2 + 4KHO = ZnCy_2,2KCy + ZnK_2O_2 + 2H_2O,$$

as already pointed out by Mr. J. S. C. Wells in a valuable paper (*Eng. and Min. Journal*, December 21, 1895). The complete reaction would then be :

$$4KAuCy_2 + 4Zn + 2H_2O = 2ZnCy_2 + ZnCy_2,2KCy + ZnK_2O_2 + 4H + 4Au.$$

The turning-point would seem to be, is hydrogen formed as these reactions require, or is it absent, as called for by the substitution-reaction? Numerous experiments have always shown that by the action of a solution of potassium aurocyanide containing only 50 mgs. of gold and zinc-turnings, hydrogen gas is set free in sufficient quantity to give a well-defined flame (4 or 5 c.c.). The hydrogen does not form in noticeable quantity at first; but as the gold begins to come down, on shaking the zinc-turnings, fine bubbles of gas escape, and may be easily collected in quantity. It may be objected that the replacement-reaction took place, and that the hydrogen was evolved by the subsequent reaction of the $2KCy,ZnCy_2$ on the zinc. Experiment shows that the double cyanide of zinc and potassium acts very slightly on zinc-shavings, but that when the latter have been partly plated with gold, hydrogen is then set free. So that the presence of hydrogen at the end of the reaction may be due to this cause, at least in part. It is also probable that the KCy is, to a certain extent, dissociated (KHO and HCy being formed). It seems likely that the nascent hydrogen will be partly absorbed by the HCy with the formation of methylamine, thus:

$$HCN + 4H = CH_3NH_2.$$

Another curious fact remains to be recorded. When zinc-turnings are placed in distilled water containing a drop of phenolpthalein no coloration takes place. When phenolpthalein is added to pure potassium-aurocyanide, no coloration takes

place. This shows the aurocyanide to be but little dissociated. When the latter solution is poured upon the former, at first no change occurs; but as the gold begins to separate upon the zinc, a deep purple stain surrounds the zinc-turnings where the gold has separated out, showing that the reaction has become alkaline. Whether this is due to the formation of caustic potash, methylamine, or the double cyanide of zinc and potassium, I have not demonstrated with certainty. The fact, however, that zinc cyanide seems to separate out early in the reaction indicates the explanation I have suggested.

Yet another point remains to be mentioned. When to a solution of potassium-aurocyanide which has been left for twenty-four hours in contact with zinc-strips, without action, a little free cyanide of potassium is added, the gold comes down at once, and the precipitation is soon complete. This fact seems an additional confirmation of the reaction that I have suggested. It will be remembered that the formation of the insoluble cyanide of zinc is a feature of this reaction. If this occurs, it is certain that a film of this substance, perhaps infinitely thin, must cover the surface of the zinc, and, in addition to the film of hydrogen, prevents the contact necessary to continue the reaction. The presence of free cyanide of potassium, of course, readily dissolves this film, and the reaction is free to continue.

The reaction which actually takes place in precipitating gold from solutions containing free cyanide of potassium will then be something like this:

$$2KAuCy_2 + 3Zn + 4KCy + 2H_2O = 2Au + 2(ZnCy_2.2KCy) + ZnK_2O_2 + 4H.$$

It will be remembered that according to the substitution-reaction, one atom of zinc replaces two atoms of gold, or 1 ounce of zinc should precipitate 6.2 ounces of gold; whereas, as every one knows in practice, 1 ounce of zinc will precipitate only from $\frac{1}{5}$ to $\frac{1}{15}$ of an ounce of gold, or thirty to ninety times less than the amount called for by the reaction by substitution. According to the reactions I have suggested, in the absence of free cyanide of potassium and caustic potash, 1 ounce of zinc should precipitate 3.1 ounces of gold. In the presence of a moderate excess of cyanide of potassium it should precipitate

2.06 ounces. The apparent discrepancy that seems still to remain between theory and practice is in reality due to the facts: first, that the free alkali (potash in particular, formed in the solution of the gold, or added to neutralize the free acid in the ore) also dissolves the zinc as potassium zincate; second, that an excess of potassium cyanide dissolves the zinc on its own account, both as the double cyanide and as the zincate of potassium; third, it should also be remembered that water containing dissolved oxygen attacks metallic zinc quite vigorously, forming hydrate of zinc.

I do not consider the reactions which I have suggested as demonstrated. I have considerable work outlined on this subject which is not yet finished, and the opinions here expressed are provisional. It is not improbable that the reaction by replacement, and the more complicated one that I have suggested may both take place under certain conditions of concentration and temperature, which are not yet understood. It is also possible that the nature of the reaction changes after the first deposition of gold.*

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* Mr. Alfred James, in a valuable paper on the "Cyanide Practice," read before the Institution of Mining and Metallurgy, London, England, May 25, 1895, calls attention to the inadequacy of the substitution reaction in explaining the precipitation of gold in the zinc-boxes. He doubts the action of KCy on the zinc, at least to any appreciable extent, and calls attention to the fact that KHO is always present in the solution, as per Elsner's reaction:

$$2Au + 4KCy + H_2O + O = 2KAuCy_2 + 2KHO.$$

He then suggests the following reactions in explanation of the precipitation of gold in the zinc-boxes:

$$(a)\ Zn + 2KHO = K_2ZnO_2 + 2H.$$

"Then, as Feldtmann has pointed out, the nascent hydrogen reduces the gold:"

$$(b)\ 2KAuCy_2 + 2H = 2KCy + 2HCy + 2Au.$$

Then he adds that a subsequent reaction takes place as follows:

$$(c)\ 2KCy + 2HCy + K_2ZnO_2 = ZnK_2Cy_4 + 2KHO.$$

When we come to study these reactions, however, it must be evident that we may combine reactions (*a*), (*b*) and (*c*) together into one reaction, for they are in reality simultaneous. We have then:

$$(d)\ 2KAuCy_2 + Zn + 2KHO = 2Au + ZnK_2Cy_4 + 2KHO.$$

It must be further evident that the 2KHO remains unaltered throughout the

However the theory of this subject may finally turn out, and even granting that, given indefinite time and zinc-shavings, it may be possible to remove all the gold from an aurocyanide-solution in the absence of free cyanide of potassium, it is still practically true that it is impossible to remove all the gold from a cyanide-solution in a reasonable time, say twenty-four hours, unless there is an excess of about $\frac{1}{10}$ or $\frac{2}{10}$ of 1 per cent. of free cyanide of potassium. This is all the more necessary in large-scale work, as fine ore-silt and the oxidizing effect of dissolved oxygen nearly always leave the shavings in the zinc-boxes more or less coated with films, which increase the difficulty of an intimate contact of the solution with metallic zinc.

8. *Recovery of Cyanide of Potassium from Strong Solutions by Zinc Sulphate.*

As objection will be made to the cuprous method of gold precipitation when strong solutions of potassium cyanide must be used, it is necessary to consider the recovery of the cyanide from such solutions previous to the precipitation of the gold.

A careful study of the ferrous sulphate method of recovering the cyanide from strong solutions, previous to the recovery of the gold, by means of copper solutions, as suggested by Prof. de Wilde, has led me to regard it as entirely impracticable in most cases that occur in practice. I have therefore sought another method besides the one already set forth (by means of sulphuric acid and aeration). This I have found in the use of sulphate of zinc. I find, however, that an American patent had been already granted, October 15, 1895, to Bertram Hunt, of Wickenburg, Ariz., for such a method of recovery. According to the patent-claims, he

"Precipitates the cyanide in the waste or spent liquor by adding a solution of sulphate of zinc containing some free sulphuric acid, whereby a cyanide precipi-

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reaction, and we may, therefore, subtract it from both members of equation (*d*). When we do this, we have:

$$(e)\ 2KAuCy_2 + Zn = 2Au + ZnK_2Cy_4.$$

In short, it appears that the explanation of Mr. James brings us back to the old substitution reaction which he started out to avoid. His reactions also fail to explain the increase of free alkali which every one recognizes as taking place in the zinc-boxes.

tate is obtained; then drawing off the supernatant liquor; then mixing with the said precipitate a quantity of sulphuric acid more than sufficient to decompose the cyanide of zinc contained in said precipitate, then subjecting the mixture to a distilling operation."

The following statements are founded on my experiments made with a solution containing 1 per cent. of free KCy and 0.1 per cent., or $603, gold per ton. When a chemical excess of sulphate of zinc is added to such a cyanide of potassium solution, a white precipitate of cyanide of zinc forms, which readily settles, and may be decanted or filtered. [If alkaline sulphides are present, they may be first removed by treating the solution by means of lead sulphate.] If less than an excess of zinc sulphate is added, the precipitate separates very imperfectly from the solution, and is difficult either to decant or filter. When the zinc sulphate contains a little free sulphuric acid, very little gold comes down with the zinc cyanide, and when an excess of the zinc sulphate is used, the precipitate separates in a manner leaving very little to be desired.

When the cyanide of zinc is acidified with sulphuric acid, and treated either, as I have already described, by pumping air through it, or by distilling it, the cyanhydric acid may be very completely recovered by collecting it with caustic potash. The latter may be entirely saturated without loss by using two vessels for condensation, the first receiving the saturated and the second a weaker solution.

A certain amount of gold will usually be found in the zinc sulphate-solution (as $HAuCy_2$). After crystallizing out the zinc sulphate to be used over again, the gold may be recovered from the mother-liquor by precipitation with sulphurous acid and copper sulphate, as before described. The amount of gold that comes down at this point seems to depend on the amount of free acid contained in the zinc sulphate. Sometimes only traces come down; sometimes nearly half of that originally present in solution.

From the filtrate from the zinc cyanide precipitate the gold may be recovered by sulphurous acid and copper sulphate, or cuprous chloride direct, as already described.

This method of treating stronger solutions, such as 1 per cent., as would be used in treating rich gold-ores, leaves very little to be desired, working on a laboratory-scale. It is in

every way superior to that described by Prof. de Wilde, witl sulphate of iron. Still, it may be questioned whether such : method could be carried out in a mining camp. There is on serious difficulty that it would meet. When the ore-solutioi contains ferrocyanide of potassium, as it is likely to do, a ferro cyanide of zinc forms, which is of the consistency of white lard This substance is hard to settle, and practically impossible t filter. It is insoluble in acids, but easily soluble in alkalies o potassium cyanide. In the presence of this substance, the sim ple acidification of the solution, followed by the removal of th HCy, by circulating air through it and condensing in causti potash, would probably be a more successful method of recover ing the cyanide.

9. *Precipitation of Aurous Cyanide by Means of Zinc Chloride.*

An American patent has been taken out on this method b Frederick Rinder, June 18, 1895. In this method:

"The solution containing gold and silver in solution is first treated by iroi sulphide, whereby the silver is removed as a sulphide; chloride of zinc is thei added to the filtrate, whereby the gold is precipitated as a double cyanide of zin and gold."

I have also investigated this method. When chloride o zinc was made by boiling an excess of metallic zinc in hy drochloric acid till the action ceased, and this was added to ε solution of potassium aurocyanide, nothing more than a trac of gold was precipitated. But when the chloride was forme by treating oxide of zinc with hydrochloric acid, the former ir excess, so that one had in reality to do with a solution of oxy chloride of zinc, 99.37 per cent. of the gold present was pre cipitated. It should be mentioned that both solutions o chloride of zinc had a slightly acid reaction to litmus-paper.

As, however, an excess of either acid or alkali dissolves th gold precipitate, this method must remain inferior to the metho of precipitation as cuprous aurocyanide; for this precipitat seems almost absolutely insoluble in dilute sulphuric acid. Th latter, therefore, possesses a sharpness and completeness tha the zinc-method can never have.

10. *The Advantages of Precipitating the Gold by Means of Cuprou Salts.*

It must be evident to those who have followed the progressiv

development of the cyanide-process that, as the method is better understood, the constant tendency is towards the use of more and more dilute cyanide-solutions. While in the beginning a solution of 1 per cent. was used, this was first reduced to one-half, then one-quarter, and finally to one-tenth, and even one-twentieth of 1 per cent. As the action of the so-called "cyanicides" contained in the ore is better understood and prevented, it seems not unlikely that the strength of the solution in potassium cyanide may be reduced to one one-hundredth of 1 per cent. or even lower. It should be remembered that much of the material treated by this process does not assay over $3 per ton, or only half of one one-thousandth of 1 per cent. gold. So that a ton of solution of 0.01 per cent. potassium cyanide solution contains 30 times as much cyanide as is needed to dissolve $3 worth of gold in a ton of ore.

The present methods of precipitation, the electrical and the zinc-shavings method, both find in these dilute solutions their great difficulty. In the electrical process the resistance of such solutions is something enormous. In the case of the zinc-shavings it is practically impossible to precipitate the gold from such a solution unless it contains one- or two-tenths per cent. free cyanide of potassium. This fact alone prevents the cyanide from being utilized to the best advantage.

In order that the cyanide should be utilized to the full, we should form the maximum of $KAuCy_2$ and leave a minimum of free KCy in the solution. This, as has been pointed out, is fatal to the precipitation by zinc-shavings. But it is just here that the cuprous method of precipitation comes into play most efficiently.

In the treatment of such solutions with a bare excess of potassium cyanide, there is no method of precipitation yet invented that can compete with it. In such a case there is not enough cyanide of potassium in the solution to bother about saving it.

The method of procedure would then be as follows: The solution would be made slightly acid by sulphuric or sulphurous acid, as might be most convenient. Then there would be added a copper sulphate solution with common salt, which had been saturated with sulphurous acid. This solution should be added until the filtered solution gives a red precipitate with potassium

ferrocyanide. The whole solution should be thoroughly stirred before this end-point is determined. A neat way to determine the end-point is to place a few drops of the stirred mixture on a double layer of fine filter-paper. On removing the upper layer, a drop of ferrocyanide of potassium will give a red precipitate of cuprous ferrocyanide on the wetted spot of the lower layer when the end-point is reached. This method avoids the delay of filtering the solution in the ordinary way. It would, of course, be best to determine the end-point beforehand, with a liter of solution, and then add the copper-salt to the mass of solution, after a preliminary calculation as to how much is required.

The solution should be allowed to stand for at least twelve hours, when it should be filtered. The filtrate should stand another twelve hours to see if any further precipitate forms; or it may be filtered first through CuS, to remove any suspended or dissolved gold, and then through old scrap-iron to throw down any copper-contents.

For the recovery of the gold from the cuprous aurocyanide, Prof. de Wilde suggests three methods as follows:

"First method: Roasting in a reverberatory furnace. One obtains thus a residue of gold and of oxide of copper (CuO). This latter is then dissolved in sulphuric acid diluted to 20° Beaumé (or in dilute nitric or hydrochloric acid), and the gold remains in the residue as pure gold.

"At the same time the sulphate of copper is regenerated, which will serve to precipitate the gold in subsequent operations,* and the same quantity of copper may continue to serve. Owing to the sharpness of the reactions, the loss of copper will be insignificant.

"Second method: Solution of the cuprous cyanide in dilute chlorhydric or nitric acid; there remains a residue of aurous cyanide which, after washing and drying, is decomposed by heat and pure gold is left behind.

"Third method: The precipitate is heated with 60° Beaumé sulphuric acid in a porcelain or iron pot; it is entirely decomposed, leaving a residue of pure spongy gold. After cooling water is added, the precious metal is washed by decantation, dried and melted. The copper has been transformed into sulphate.

"The first method appears to me the most rational, the roasting being attempted once or twice a month only. It is an inexpensive operation, and the sulphate of copper is thus regenerated."

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* "The sulphate of copper thus regenerated should be crystallized by cooling the solution, and the crystals drained from the adherent acid mother-liquor. A solution of sulphate of copper containing a notable quantity of sulphuric acid is not adapted to the precipitation of gold. The mother-liquors, after being strengthened by the addition of sulphuric acid, serve very well for the attack of the mixture of oxide of copper and gold."

In this matter I agree with Prof. de Wilde. After being carefully dried the conversion of the cyanide takes place very quietly at a low red heat, and the spongy, porous, black residue readily dissolves in the sulphuric acid, leaving the gold very clean. Care should be taken not to alloy the gold and copper by a reducing atmosphere and too much heat.

A fourth method would be to dissolve both gold and copper cyanide in a strong KCy solution, and precipitate pure gold by the dynamo. With less than 2.5 volts and a strong solution of KCy this is possible, the copper remaining in solution.* This I have verified. All the objections to electrolysis apply, except that the bulk of the solution would be small, and it would be concentrated.

In many cases it would probably prove more advantageous for the reduction-works to ship this precipitate without attempting to reduce it, as the technical skill to do this occasional work is hard to get in mining camps.

The methods here outlined will certainly fail in the hands of those without chemical knowledge and engineering skill, and many unforeseen difficulties will probably have to be overcome before they can be utilized in practice. Nevertheless, I feel very confident that in some of the methods here outlined for the precipitation of gold by means of cuprous salts will be found the missing link in the chain of operations necessary to utilize the extremely dilute solutions of cyanide of potassium, which have been found effective in extracting gold from low-grade ores. If this should prove to be the case, and the usefulness of the method should be extended, particularly in California, my native State, I shall feel amply repaid for this long labor.

III.—The Treatment of Ores.

I have here reached the limit laid out for myself in the present paper. Still a few words on the application of the process to ores may not be out of place. In the first place, it should be said that the usual extraction from the South African tailings averages only from 60 to 70 per cent. While such results on low-grade products that can be treated in no other manner may be eminently satisfactory, they should be regarded as deci-

* H. Freudenberg, *Zeitschrift für Physikalische Chemie*, xii., p. 97, 1893.

dedly unsatisfactory for rich ores and concentrated sulphurets, assaying perhaps $100 per ton or upwards. It is to the treatment of such products, particularly the latter, that I have given the most attention. Although at first I could obtain extractions at most averaging not more than 70 or 80 per cent., I was finally able to reach uniform extractions that were eminently satisfactory with quite a wide range of ores. Thus, with the concentrated sulphurets from the Idaho Mine, Grass Valley, assaying $120 per ton, I began with 70 per cent. and finished with 98 and 99.45 per cent. With the Alaska Treadwell concentrates, assaying $50 per ton, I began with 75 per cent. extractions, and finished by extracting all but a trace. In a lot of concentrates from the Bald Eagle Mine, Alaska, assaying $280 per ton, I was finally able to extract up to 97 per cent. On the other hand, with a lot of sulphurets from the Bay State Mine, Amador county, California, assaying $160 per ton, 90 per cent. is the best result yet obtained.

Each one of these ores demanded a long and painful study before these results could be obtained with any certainty, and there still remain some difficulties to be overcome before they can be reached on the large scale. Still there seems a strong probability that this will be accomplished. And while it is unlikely that the cyanide process will displace any of the standard methods of gold-extraction it will certainly supplement them where they are weak, and will perhaps enable low-grade ores to be worked that will not now pay for chlorination.

I cannot close without a warning to those who expect extravagant results from this method in the hands of persons utterly ignorant of chemistry. No process was ever introduced requiring a more thorough and subtle chemical knowledge, and without it success is absolutely impossible. But engineering skill and experience are also equally indispensable, and success can only follow where these agents co-operate to produce the final result.

Some Mines of Rosita and Silver Cliff, Colorado.*

BY S. F. EMMONS, WASHINGTON, D. C.

(Colorado Meeting, September, 1896.)

THE history of the mining region of Custer county has been somewhat peculiar. Although, in the broader features of geological structure, it bears a strong resemblance to its newer and now more famous neighbor, 40 miles to the northward, the Cripple Creek mining district, it has been noted for failures rather than for successful mining ventures. Yet it has a number of mines that have proved themselves to be remarkably rich, and many others that have well-defined veins and very rich ore, though they never have been worked to any great depth. A few of the deposits seemed to the early miner so anomalous in form and manner of occurrence that they were popularly described as "contrary to all geological laws." The region has had several "booms," succeeded by more than usually violent reactions, the latter having proved more permanent than the former. It was, for a long time, the favorite resort of patent-process mongers, and the ground is now strewn with relics of various unsuccessful reduction-plants. Mismanagement, ignorance and disagreements among mine-owners seem to have been quite as responsible for the want of success in the region as the quality of the ores or the form of the deposits.

In the early eighties, when the booms had not entirely died out, a division, then under my charge, of the U. S. Geological Survey, undertook an examination of the region for the purpose of making a study of its ore-deposits. The areal survey was carried to completion by Mr. Whitman Cross, but the study of the mines could never be completed for the reason that most of them were closed, for one reason or another, before we had time to visit them. From time to time, one or another of the important mines has been started up again, but it was not

* Published by permission of the Director of the U. S. Geological Survey.

always possible to visit the region while they were still open, and hence the economic study of the region has never been completed. Such facts as I have been able to gather from personal observation and from other sources have been finally put together in an article published in the last *Annual Report* of the Director of the Survey, of which the present paper, descriptive chiefly of the four principal mines of the district, is a condensation. Although necessarily incomplete, it contains some facts that will, I think, prove of interest to the members of the Institute.

Topography.

The region containing the mines is situated on the western slope of the Sierra Mojada or Wet Mountain range, about 25 miles southwest of Canyon City, and extends from the fertile bottoms of the Wet Mountain valley about 10 miles diagonally up the slope toward the crest of the range. The town of Silver Cliff is situated on the Quaternary slope, at the southern end of a low plateau of rhyolite, and at the base of the cliff from which its name was taken. Rosita, at the other extremity of the area, nestles in a little valley among the so-called Rosita hills. These hills cover about 25 square miles, and are made up of volcanic rocks, which have broken up through, and now cover more or less completely, a basement-complex of gneisses and granites, hitherto considered to be of Archæan age.

About 3 miles from the western base of the Rosita hills there rises, out of the Quaternary that slopes gently westward toward the bottom-lands of the Wet Mountain valley, a sharp conical peak known as Round mountain. West of this stretches a plateau, nearly 2 miles square, ending at the north in a group of low hills, called the White hills, from the rock of which they are composed.. At the southern edge of the plateau is a cliff, nearly 100 feet high, in which silver has been found, and from which the mining town of Silver Cliff at its base was named. North of the White hills is another and somewhat higher group of hills, known as the Blue mountains.

Discovery and Development.

The first discoveries in the region were made by ranchmen of the valley while hunting their stray cattle. Prospectors

came first to the Rosita region in the autumn of 1872. The most prominent and noted deposit in this vicinity was the Humboldt-Pocahontas vein, which was discovered in April, 1874, and worked almost continuously for about 13 years. The town of Rosita reached the height of its prosperity in 1875 to 1877.

The second great mine was the Bassick, which was discovered, about 2 miles north of Rosita, in 1877. This has been the most important producer of the region, having been early bought by New York parties and capitalized at $10,000,000. It has been closed down since 1885, having produced about $2,000,000 in gold and silver.

The next important discovery, that of the Silver Cliff deposit, in 1878, gave rise to a greater mining boom than the region had yet experienced. This was, in part, due to the general mining excitement throughout the country, consequent upon the opening of the Leadville mines, but in greater part to the belief, at first entertained, that the whole plateau was a mass of silver-ore.

The Bull-Domingo mine was discovered about that time in the Blue mountains, and the sale of these two properties in the east, with their capitalization at $10,000,000 each, was the culminating phase of the boom, which thereafter began to die out.

Geological History.*

With the exception of a very local and probably recent lake-bed, no sedimentary formations appear in the region. Upon a basement-complex, consisting mainly of gneisses cut through by granite, with some dikes of syenite and peridotite, a series of eruptive rocks has been poured out, forming hills similar in general form to those of the present day, but probably more elevated, since the tendency of modern erosion has been to plane down the hills and fill up the valleys. The filling-up process was going on during the time of active eruption, as will be shown below in the description of the Geyser mine. The fossil leaves found in the rhyolite mud-flows of the Rosita hills

* A full description of the geological structure of the region, with geological maps and sections, will be found in the 17*th Ann. Rep.* of the Director, U. S. Geol. Sur. (1895–6), Part II., in a paper by Mr. Whitman Cross on the "Geology of Silver Cliff and the Rosita Hills."

render it probable that the volcanic action occurred during the early Eocene. Mr. Cross thinks that the various rocks which now make up the Rosita hills must be regarded as a series of products from the same volcanic or eruptive source. Most of them were surface-effusions, but there are also dikes, and the present relationship of the different bodies is due, in a few instances, to faulting. The eruption of two of the rocks was plainly begun by explosive action, producing much fragmental material. The other effusions were massive eruptions, producing even-grained rocks, almost identical with the forms shown in the dikes.

The principal eruptive rock-varieties, with their local names, are, in the order of their eruption, as follows:

Rosita andesite; Bunker andesite; Fairview diorite; Bald Mountain dacite; rhyolite; Pringle andesite; trachyte; Bassick agglomerate; mica-dacite; limburgite (a very basic basalt).

As Mr. Cross shows, there was probably an active volcano somewhere near the present site of Rosita, which emitted lavas, alternating with ashes and fragmental material. The first eruption was that of the Rosita andesite, which was poured over the pre-existing Archæan hills and valleys. It is mica-hornblende-andesite, as a rule much altered. In the valley of Rosita creek it is characteristically a purplish- or bluish-gray breccia; near the mines it is often soft and somewhat bleached.

This was followed by the eruptions of the Bunker andesite, the Fairview diorite, the Bald Mountain dacite and the rhyolite, none of which, however (except a few outlying dike-like bodies of the rhyolite) appear in the neighborhood of Rosita.

These were succeeded by the quiet eruption of the Pringle andesite, which must have covered the whole southern slope of the hills, and of which a remnant still covers a great part of Pringle hill, west of Rosita. This rock is relatively fresh, yet sometimes kaolinized.

The next eruption was that of the trachyte, which now forms the cap of Game ridge, and is seen in a series of dike-like bodies, cutting the Pringle andesite as well as the earlier rocks. The body on Game ridge fills an earlier (probably rather shallow) depression; but its present outlines are determined in considerable measure by later faults. The location of its source does not appear; but it may well have been connected

with some one of the dike-like bodies of the same rock. The rock of the dikes is dark-gray porphyry, showing glassy sanidin when fresh.

Of the still later rocks above mentioned, nothing further need be said here. Their occurrence is described below.

The ore-bearing veins cut the trachyte, thus proving their later origin. In most of the faults traversing the trachyte some mineralization may be observed; and many of the ore-discoveries around Game ridge have been fault-fissures. Such are the Horton and Hard Cash mines on the California fault, and the Nellie and Sleeping Pet, on the Nellie fault, which are between trachyte and gneiss. On the Twenty-Six fault, which is partly in trachyte and partly between Rosita andesite (on the north) and trachyte (on the south), are the Summit, Polonia and Twenty-Six mines. This fault exhibits a complex of parallel fissures, with crushed and shattered material between. In the vein itself are boulders of andesite-breccia and of granite rounded by attrition.

The Humboldt-Pocahontas Vein.

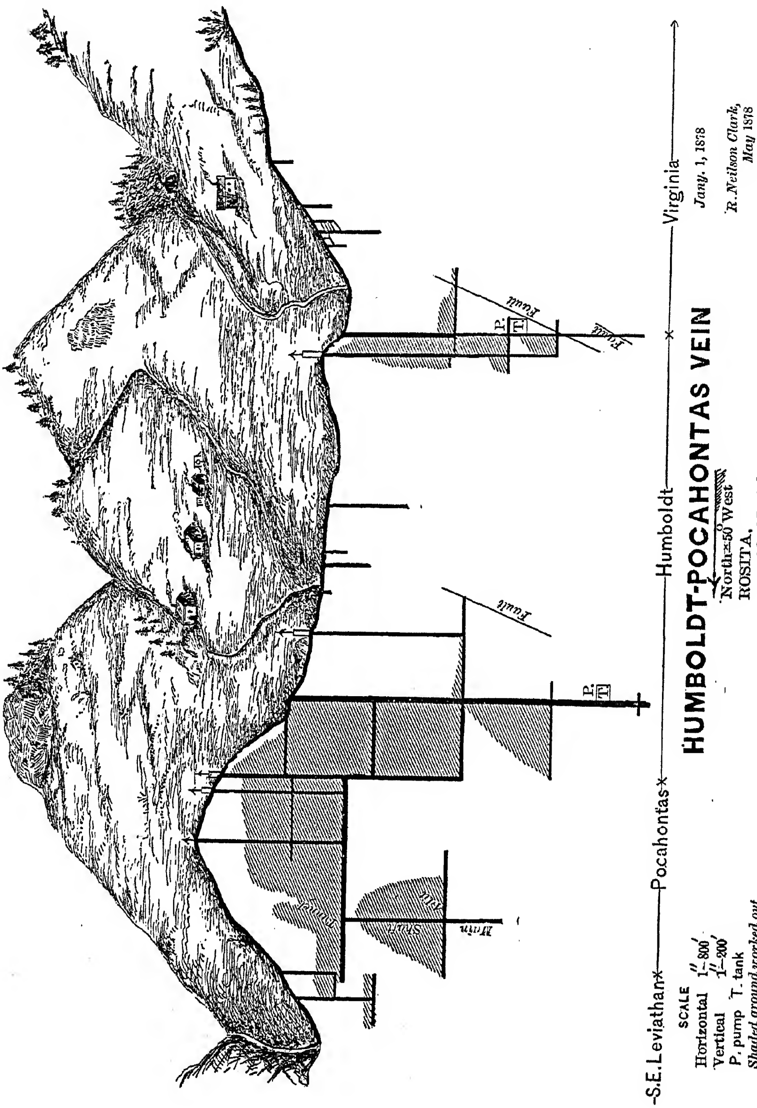
This is an excellent example of what is generally called a true fissure-vein. It is remarkably regular in direction and dip in its upper levels, and has contained, in the length of about 4000 feet along which it has been opened underground, a somewhat unusual amount of rich ore. An excellent account of the workings up to 1878 in the various mines upon this vein was given by the late R. Neilson Clark in his paper on the subject,* from which I reproduce in Fig. 1 a profile, showing the division of the vein among the several companies, and also the position of certain features, interpreted by Mr. Clark, upon the evidence accessible at that time, as faults. For a complete description of the vein and its development, the reader should combine what is here given with the earlier account of Mr. Clark, from which, however, I take a few facts essential to the comprehension of my own statements.

The strike of this vein is N. 50° W., and it dips 60° to 72° S.W. (away from the hills). The ore is mainly tetrahedrite, carrying copper and iron pyrites, some galena, stephanite, and

* *Trans.*, vii., 21.

other antimonial silver-minerals. Barite is the principal gangue-mineral, calcite being present in subordinate amount. The

Fig. 1.



vein had produced up to May, 1878 (principally from the Humboldt and Pocahontas mines, the shipments of the Southeast Leviathan and the Virginian having been comparatively insig-

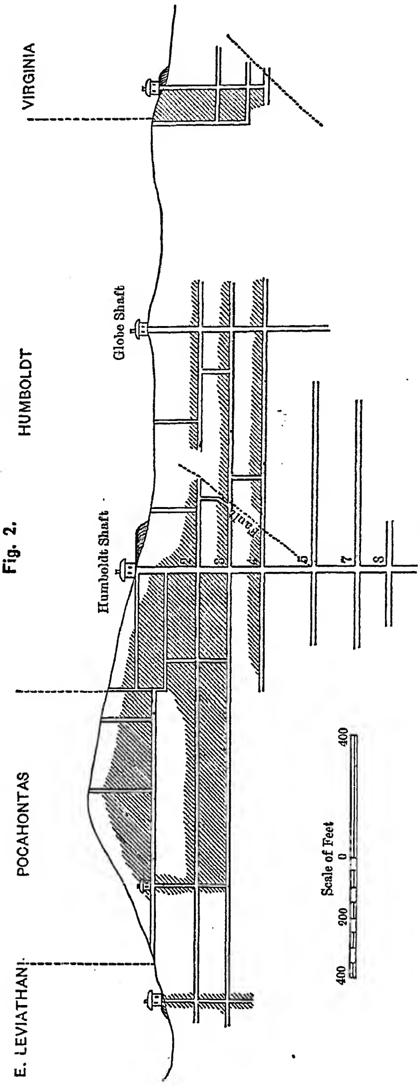
nificant), 4893 tons of ore shipped, of a gross value of $565,929, or $115.56 per ton, in currency.

The Pocahontas and Humboldt mines were examined by the writer in 1883, and were visited again in 1887. Fig. 2 shows the position of the shafts, and, in a general way, the portions of the vein which had been found, down to the latter date, to carry rich ore. The latest workings, however, are not shown. In 1887 the Pocahontas mine was worked at the 300-foot level through the Humboldt shaft; and it is not known whether the vein has since been worked to greater depths in the Pocahontas ground. Above this level the vein was remarkably regular and straight in that ground, striking N.W.–S.E., with an average dip of 45° S.W. The country-rock on either side is Rosita breccia, sometimes, however, losing its brecciated structure, and showing, especially on the hanging-wall, solid lava only. The breccia is often quite dark and hard; at other times, bleached and decomposed. The hanging-wall is especially smooth and regular, generally with a thin clay-gouge. The vein-matter is for the most part decomposed country-rock, impregnated with pyrite, chalcopyrite, tetrahedrite and some silver sulphides and antimonides associated with barite, often in fine tabular crystals. Its width varies from a few inches to 2 feet. The foot-wall is wavy, and has no clay-gouge, though it forms a comparatively distinct separation between ore and country-rock. The pay-ore is said to have run in horizontal streaks, and to have been continuous from one end of the Pocahontas claim to the other.

In the Humboldt ground the vein is of the same general character, but its dip varies from 45° to 70°. The shaft has followed the average dip (60°) to the fourth level, where the vein splits. Boulders of granite and gneiss could occasionally be observed completely inclosed in the andesite country-rock. Down to the fourth level (400 feet on the dip) the vein is comparatively regular. Below that level a horse of country-rock about 30 feet wide divides it into two. The northeastern vein gradually grows steeper and becomes reversed, taking a dip to the north. For some distance it is said to have gone down in a series of steps that were alternately very shallow and very steep. From the fifth level down the northeastern vein had granite on the hanging or northeastern wall, and on the other

wall breccia gave place to porphyry. The ore became mostly pyrite of rather low grade—20 to 30 ounces of silver per ton.

Fig. 2.



HUMBOLDT-POCAHONTAS VEIN.

The southwestern or main vein, on the other hand, grew shallower in dip, and diverged from the other until at the eighth level they were nearly 800 feet apart. The cross-cuts

between the two were in broken rock until the seventh level was reached, when they showed solid porphyry, with a regular sheeting parallel to the southwestern vein. The northeastern vein gradually died out, and the southwestern or main vein was thought at one time to be lost, but was recovered on the eighth level. Here the breccia had disappeared, and the same even-grained porphyry appeared on both walls. In the vein were found, here and there, rounded fragments of red granite of various sizes, sometimes entirely coated with gray copper. In one case the granite fragment was so large that it took a long time to cut a drift through it.

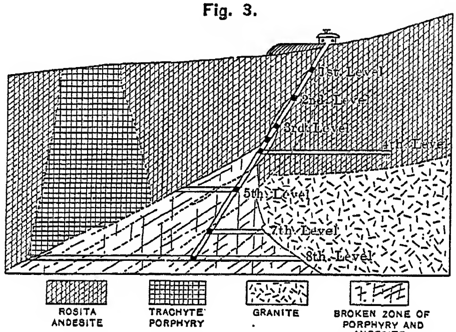
The facts with regard to these lower workings were obtained in 1887 from the manager, Mr. Thornton, an extremely careful and intelligent observer. All levels below the fifth were, at that time, full of water, having been abandoned by the owners on account of the low grade of the ore, and difficulties of mining incident to the greatly flattened dip. It is sincerely regretted that the lower parts of this vein could not have been examined in detail, since what is reported concerning the deeper portion near the Humboldt shaft indicates an opportunity of studying here the root of an extensive system of vein-fissures. It has long been the opinion of the writer that the idea, generally accepted among miners, that, because an ore-deposit has been formed on what may be called a true fissure-vein, it necessarily has an indefinite extension in depth, is much in the nature of a popular fallacy, and that the extension in depth of ore in a fissure is as likely to terminate within a measurable distance as the extent of ore-deposition on what are generally called "blanket-deposits." The present case seems to illustrate this idea as far as the facts enable one to judge; for while the fault-fracture constituting the vein has been traced horizontally for more than a mile, yet, where it has been explored in depth, it splits, and shows signs of dying out altogether at 800 feet from the present surface. To this depth should be added, for the original vertical extent of the fault-fracture, the vertical depth of the material carried away by erosion since the fissure was formed. Concerning this depth, there are but few facts available for a valid deduction, but, in all probability, it can hardly have been much over 1000 feet.

It is seldom that an important fault-movement takes place on

absolutely a single plane; and in this case there have been evidently one or more parallel planes of movement—which have not been thoroughly explored, probably because the main vein was so regular and well defined; for the miner generally likes to follow a well-defined wall, and is reluctant to go behind it. In the present case it is possible that more ore might have been found on one of these secondary planes in the upper workings.

The vein appears to split into a series of conjugated fractures in depth in the Humboldt ground as it reaches the porphyry (trachyte?) and granite. The diagrammatic section given in Fig. 3 is founded on too few facts of observation to

Fig. 3.



SECTION THROUGH HUMBOLDT SHAFT.

be claimed to be accurate, but it represents a probable manner in which the main fault-fracture splits up in depth.

In the case of the Comstock and some other important veins, the vein-fissure has split and opened out upward toward the surface. For this reason, miners are apt to expect that two nearly parallel veins which are in proximity at the surface will be likely to come together in depth. In the present case the conditions are reversed, showing that the splitting is not necessarily upward in every case. If the ore-bearing currents were ascending along a set of fissures, such as is represented in Fig. 3, it is readily conceivable that the richest and most abundant ore would be concentrated in the upper part after the converging fissures had united.

The Bassick Mine.

Mount Tyndall is an extremely picturesque conical hill, about

600 feet high, at the N.E. extremity of the Rosita eruptive area. On Bassick hill, a shoulder upon its southern slope, occurs the outcrop of the Bassick ore-body. This body is enclosed in an agglomerate, mainly of andesitic material, which forms the whole mass of Bassick hill and a considerable proportion of Mount Tyndall, and which, as it contains fragments of both the Rosita and the Bunker types of andesite, must be more recent than either. The top of Mount Tyndall is a dense banded rhyolite, of still more recent date, which has protected the agglomerate, otherwise easily softened and eroded. On Bassick hill, however, the agglomerate has been protected by its own silicification in the vicinity of the ore body. In the upper knoll, in the ridge connecting Bassick hill with Mount Tyndall, occurs a dike of dense black rock, 3 to 4 feet thick, described as limburgite, in which fresh olivine crystals are the only phenocrysts. This dike is of special interest, because, as Mr. Cross shows, the Bassick ore-body must have been formed before its eruption.

Of the fragments in the agglomerate, some are rounded; others, subangular; few, if any, distinctly angular. They vary in size from that of gravel-stones to blocks 3 feet in diameter. In the quarry above the Bassick mill, where the exposures are good, the matrix is white, often homogeneous in appearance, sometimes hard and flinty, sometimes earthy and crumbling. Where the residual products, silica and kaolin, are locally concentrated, it sometimes forms a conglomerate.

Mr. Cross gives sections showing the assumed form of the agglomerate body and of the volcanic neck which it is supposed to fill. Of the actual outlines below the surface there is little evidence, but volcanic necks in general have nearly vertical walls, and, at the depth of 1400 feet in the Bassick, this agglomerate shows no signs of the proximity of the Archæan.

An excellent description of this mine was given by Mr. L. R. Grabill in 1882,* at which time a depth of something more than 800 feet had been reached by the shaft. The statements here made concerning the mine above that level are based on Mr. Grabill's paper, which states that the fissure is an irregular opening, nearly elliptical in horizontal section, 20 to 30 by

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* *Trans.*, xi., 110, "On the Peculiar Features of the Bassick Mine."

nearly 100 feet in size, and extending downward vertically, with slight windings, nearly 800 feet. No walls or distinct boundaries between ore and country-rock have been found. The ore occurs in "scales" or layers surrounding the porphyry (andesite) boulders of the agglomerate. Near the center of the body, the size of these rounded fragments is greater, leaving larger interstitial spaces for the combs of mineral, and the scales are there both thicker and richer in precious metals. The boulders or pebbles decrease in size, and the scales become thinner, as distance from the center increases, until the scales, and finally all traces of the precious metals, disappear.

The minerals noted by Mr. Grabill were sphalerite, jamesonite, tetrahedrite, smithsonite, calamine, free gold, gold and silver tellurides, chalcopyrite, pyrite, quartz and kaolin—no barite, calcite, or other spars. These minerals were deposited in concentric shells around boulders from 10 to 600 mm. (average, 100 to 300 mm.) in diameter. The shells are usually three, sometimes four, in number and occur in the following succession, from within outward:

1. A hard black layer, from a hair-line up to 5 mm. thick, of sulphides of zinc, antimony and lead, usually carrying about 60 ounces of silver, and 1 to 3 ounces of gold per ton.

2. A layer of lighter color (not always distinct), richer in lead, silver and gold (sometimes 100 ounces of gold, and 150 to 200 ounces of silver, per ton).

3. A shell of beautifully crystalline sphalerite, 5 to 50 mm. thick, carrying usually 60 to 120 ounces of silver, and from 15 to 50 ounces of gold per ton, and constituting the principal source of value in the mine. It shows also a considerable amount of iron and some copper. This is often the outer coating and is rough on the outside, by reason of the projecting points of the crystals of sphalerite.

4. The next shell, when there is one, is chalcopyrite, sometimes only as sparse crystals scattered over the sphalerite, sometimes in a layer 10 or 20 mm. thick. It carries up to 50 or 100 ounces of gold, and the same of silver per ton.

5. Outside of this there is sometimes a thin coating or sprinkling of pyrite crystals. Surrounding all, and filling the remaining spaces between the boulders, is kaolin.

Of the above, No. 1 is always present, being the innermost

coating of the largest boulders and the sole coating of the smallest pebbles. On the other hand, the sphalerite shell is wanting in the outer part of the ore-body. Calamine, smithsonite and most of the free gold are found above the water-level. Tetrahedrite never occurs as a shell, but always fills vacancies outside the coated boulders, and is intermingled with quartz, and sometimes with broken pieces of the first and second (never of the third and fourth) coatings. In the same mass the tellurides of gold and silver are contained. Quartz occurs in the open spaces between boulders, never in the coatings. It is mostly amorphous, and ranges in color from opaline white, through gray, blue and brown, to black.

Charcoal has been found both within and without the ore-shoot, down as far as 765 feet. The amount found at this level would make a cubic foot. It was most common near the water-level. It is mostly soft and friable, though still showing the grain of the wood. Some is much silicified. Nearly all has its pores filled with glittering crystals of pyrite. One piece, 30 to 40 mm. long, showed the cross-section of a trunk or branch, 40 to 50 mm. in diameter.

Mr. Grabill is unable to account for the fact that the boulders are rarely in contact with each other, as he does not see what could have supported the fragments, or kept them apart, while the metal-bearing shells were being deposited around them.

In our later examination (of levels opened after Mr. Grabill wrote his paper) neither gold nor the oxidized minerals nor the tellurides* were observed. The more common minerals noted were pyrite, sphalerite (both the dark ferruginous and the yellow resinous varieties), galena and chalcopyrite.

In some cases the different bands were separated by a thin earthy band carrying some calcite or dolomite, which often con-

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* This does not impugn the accuracy of Mr. Grabill's observations as to the upper levels. In further confirmation of his statement as to the character of the ore, Mr. Richard Pearce informs the writer that the Argo works were the first to purchase Bassick ore, and that he found tellurium in it. The following analysis of a piece of rich Bassick ore was made by Mr. F. C. Knight, chemist of the works: Gold, 1.64 (or 475.32 ounces per ton); silver, 2.38 (or 694.15 ounces per ton); copper, 17.43; zinc, 18.19; lead, 10.18; iron, 7.96; bismuth, 0.56; tellurium, 2.72; sulphur, 26.07; arsenic, 1.90; antimony, 10.20; total, 99.23 per cent. This piece of ore was apparently made up of tetrahedrite, chalcopyrite, galena, sphalerite, and possibly petzite.

tained a flat, open space, parallel with the banding. The elongated vugs found sometimes in the boulders of country-rock are generally lined with minute crystals of calcite or dolomite, but where one of these vugs is cut off by the metallic band around the boulder a thin film of pyrite lines the vug near its mouth, which is in turn covered by bluish chalcedony that lines all the interior parts of the vug.

The boulders within the area of ore-deposition are generally bleached and decomposed; they have lost almost all traces of their basic silicates, and their feldspar crystals are thoroughly kaolinized. On the other hand, they are very freely impregnated with pyrite, which, coarser and more abundant near the periphery, decreases and becomes finer grained toward the interior.

In the barren breccia surrounding the ore-body, the interstitial spaces between the larger fragments of andesite are filled by a bluish, siliceous mass which incloses, also, smaller fragments of the country-rock. It is in part a finely granular mass, in part compact, approaching chalcedony. The latter contains no metallic minerals; but fine-grained pyrite is scattered through the granular portions, and still more abundantly through the interior of the fragments, where it seems preferably to replace the basic silicates, but never completely to fill the spaces originally occupied by them.

Although there is no definite boundary between the ore and the barren breccia, it was observed that there were several fracture-planes or joints in the agglomerate, cutting each other in such a way as to rudely outline the form of the ore-shoot.

As the mine had been permanently closed down before a second visit to the district was made, no personal observations of the lower levels were possible. From the superintendent and foreman the following facts with regard to them were, however, obtained: When the shaft was down to the tenth level, a second ore-body or shoot was struck to the east of the main body and the two ore-bodies were thought to be approaching each other. The ore in the new body carried rather more zinc-blende than the first shoot; and below the tenth level the ore formed a thinner scale about the boulders, and was supposed to contain a considerable amount of tellurium.

In the 1200-foot level (below the tunnel) the ore was 4 inches

thick around the boulders, and the new shoot was 150 feet from the old one, but not developed at the time of the suspension of work.

Fig. 4 is a somewhat diagrammatic section of the two ore-bodies on a line running east and west through the shaft. It was obtained from the intersections of the sets of timbers that fill the old stopes at each of the levels. These, of course, represent only the limits of the pay-ore, and that not always with perfect accuracy. Hence the main merit of the section is to show the general form and direction and the approximate limits of the pay-ore.

Genesis of the Ore.

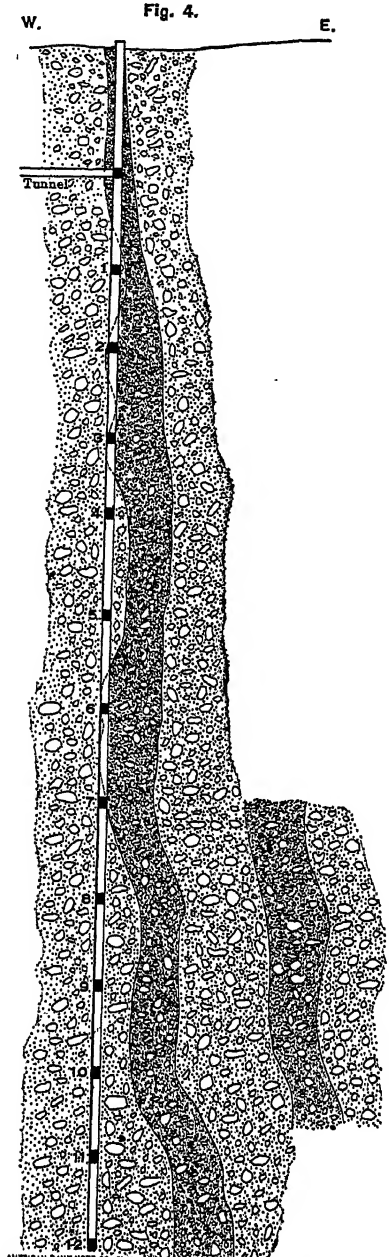
The data obtainable, though meager, establish two series of phenomena connected with this deposit which point decidedly to an origin and manner of formation differing from that of the ordinary ore-deposit. These are:

1. The physical character of the body: Its form of a long, slender, nearly vertical chimney, in an agglomerate which appears to have filled the neck or crater of an old volcano.

2. The fact that in the composition of its ore the earthy minerals, such as barite, calcite, and quartz, which are common in the other ore-deposits of the region, do not form an essential part of the vein-material, but occur, if at all, only in subordinate amounts in the actual ore-channel, and are apparently secondary ingredients, introduced after the deposition of the other minerals.

The conclusion from these facts seems inevitable, that the ore-deposition was here a phase of the volcanic eruption—not, however, as has been suggested by some, that it took place during the active eruption of the volcano, and that the metallic vapors were condensed around the rock-fragments as they were thrown upward and fell back again into the neck of the crater (for under such circumstances they ought to have impregnated the whole mass of the agglomerate), but rather as a phase of the dying activity of the volcano, after all explosive action had ceased, and when the agglomerate had cooled and become consolidated. It would seem that here, if anywhere, was presented a typical instance of a deposit due to fumarolic action, which is the explanation French geologists are wont to give to most

Fig. 4.



SECTION THROUGH BASSICK SHAFT.

deposits of metallic minerals in close connection with igneous rocks—an explanation which the writer has been unwilling to accept for most of the deposits he has studied.

As a result of the studies of gaseous emanations from the lavas of active volcanoes, made mostly by French geologists, the following phases in fumarolic activity are recognized:

1. *Dry or Anhydrous Fumaroles.*—In these the gases issue quietly from the fused lavas at a very elevated temperature (500° C.) in the form of white fumes. They consist mainly of anhydrous chlorides. Chloride of sodium is most abundant (up to 94.3 per cent. at Vesuvius); next chloride of potassium (up to 16 per cent.); and small amounts of the chlorides of manganese, iron, and copper have been detected.

2. *Acid Fumaroles.*—These issue from the sides of the lava-stream at some distance from the still fused lava, and are at a less elevated temperature than the preceding (300° to 400° C.). They contain a mixture of sulphurous and hydrochloric acids, with enormous quantities of steam (about in the proportions respectively of 1 to 10 to 1000). It is assumed that the temperature is not sufficiently elevated to remove the chlorides of the alkalies from the lavas.

3. *Alkaline (or Ammoniacal) Fumaroles.*—These are characterized by the presence of chlorohydrate (possibly also of carbonate) of ammonia. They consist mainly of aqueous vapor, and contain, also, a little sulphuretted hydrogen; and the temperature of the gas is about 100° C.

4. *Cold Fumaroles.*—These consist almost entirely of aqueous vapor, and have a temperature below 100° C. They contain sulphuretted hydrogen and about 5 per cent. of carbonic acid, and might be designated sulphydric fumaroles.

5. Finally the *mofettes*, or emanations of carbonic acid, mark the close of the eruption. All these vapors contain atmospheric air, oxygen, and nitrogen; also hydrogen, and sometimes hydrocarbons.

More briefly, the various phases in gaseous emanation may be characterized as yielding (1) alkaline chlorides and hydrochloric acid; (2) steam and sulphurous acid gas; (3) carbonic acid gas.

The products that have been observed in the vicinity of lava-flows that may be ascribed to fumarolic action are: Specular

iron, chloride of lead or cotunnite, boric acid, and the sulphides of arsenic (realgar and orpiment).

As both the gases and their products mentioned above have been formed under ordinary atmospheric pressure and with practically free access of atmospheric air, we are by no means justified in assuming that the same conditions prevailed in the conduit in which the deposit under consideration was formed; since part of it is, even now, nearly 1500 feet from the surface, and in all probability nearly as much of the upper part has been eroded away.

The gases directly emitted from the fused lavas are, as remarked above, anhydrous. But it is not conceivable that such mineral products, and in such forms as are found in the Bassick mine, could have been produced by dry distillation. Indeed, the great French geologist, Élie de Beaumont, who was among the first to insist on fumarolic deposition, himself admitted that such deposition must have been through the agency of water, and could not have been a dry distillation. The metallic minerals in this ore-body were evidently deposited mainly as sulphides and, to a limited extent, as tellurides; it seems, therefore, more reasonable to assume that they were formed during the closing phases of fumarolic activity, when H_2S and SO_2 were the prevailing gases, if, indeed, it was in the gaseous form that they were concentrated in their present locus. It is not impossible, however, that the aqueous vapors carrying the sulphides and tellurides of the minerals were under so great a pressure, at the depth at which most of the deposition took place, that if their temperature was not very much above 100° C., they were condensed into liquid form.

Our brief examination of the ore-body showed two or more lines of fracture in the agglomerate, intersecting each other near the outer limits of ore-deposition. Moreover, it is said that the second ore-chimney, which has only been observed as yet in the lower levels, is on the same line of fracture or jointing that runs through one side of the main body. It would appear, therefore, that it was the intersection of certain fracture-planes that determined the course of the ore-bearing channel, and that the ore-body is not necessarily the center of the volcanic vent, but that, as a second ore-chimney has already been discovered on one side of the first, it is by no means impossible that other

chimneys or ore-shoots may exist in the mass of the agglomerate, and might be discovered by judicious and systematic exploration in the direction of the principal fracture-planes.

It will be highly important to students of ore-deposition to obtain accurate and detailed information with regard to this deposit, especially if it should be explored here to the great depth contemplated in the Geyser mine. For such an exploration, there is certainly as great a promise of profitable return here as there.

Singularly, little is known of deposits to which a gaseous origin can, with certainty, be assigned. None have come under the observation of the writer previous to this, and, as has been said above, he is inclined to admit the possibility that the ore-deposition here was made from superheated sulpho-aqueous solutions which, under ordinary atmospheric pressure, would have assumed a gaseous form. Mr. Grabill failed to see how the shells of mineral could have inserted themselves between adjoining rock-fragments or boulders when the latter were actually in contact, because, at the time he wrote, the capabilities of replacement-action had not yet been so far demonstrated as to cause it to be recognized, as it is to-day, as a common form of ore-deposition. At the time of deposition, the agglomerate, in what is now the ore-channel, was probably in no essentially different condition from that in which the country-rock around the ore-body is now, except that it may have been somewhat shattered along the fracture-planes that are supposed to have determined the course of the ore-bearing currents, whether of gas or liquid. The interstices between the fragments must, therefore, have been more or less completely filled by the finest tuff-material; and this was removed as the metallic minerals were deposited, the ore-bearing materials eating through this more or less porous material until they reached the fragments or boulders of compact lava, when they were precipitated on the periphery of these boulders. Except for the absence of earthy gangue-minerals, the results of this deposition differ in no way, as far as could be observed, from those that have been noted in deposits which were undoubtedly made from aqueous solution, and the resemblance extends to the more complete rounding of the rock-fragments, which is an almost invariable result of the action of aqueous solutions.

THE BULL-DOMINGO MINE.

The Blue mountains are a group of isolated hills rising out of the gently sloping plains on the west flanks of the Wet Mountain range. Evidently they once formed part of the original Wet Mountain plateau, or peneplain, and have resisted erosion to a greater degree than the plains around them through the more enduring character of the rocks of which they are composed.

They consist of a central ridge of augite-gneiss of massive texture, with two curving ridges branching off from the southern end, the western of which is of the same rock, while the eastern is more clearly banded, and passes into hornblende and biotite-gneiss. In the depression between these and the central ridge is a white muscovite-gneiss, with neither hornblende nor biotite. The foliation has a prevalent strike between N. 40° E. and N. 75° E., with a dip of 40° to 90° to the northwest. In the northern part of the hills this strike bends more to the northwest. Dikes of granite, syenite and diabase traverse these rocks. The former generally runs parallel to the banding or foliation of the gneiss.

At the southern end a ravine, running southward, splits the group into two rather unequal parts. On the spur west of this ravine is a narrow dike of red fine-grained syenite running about N. 25° E. and dipping 79° to the southeast. A short distance to the east of this dike is a somewhat thicker dike-like body of pink granite, which has strike to the northeast, with the foliation of the gneiss, and which dips 79° to the northwest. Within the angle formed by these diverging dikes, and at the foot of the steeper slope of the hills, was the outcrop of the Bull-Domingo ore-body, now a round, bowl-shaped hole, 70 to 100 feet in diameter, and perhaps 30 feet in depth.

Mode of Occurrence of the Ore.—The Bull-Domingo ore-body has been generally classed with that of the Bassick mine as a deposit in a volcanic neck, a very unusual form of ore-deposition. The ore itself, consisting of concentric shells of brilliant crystalline galena and spar, forming coatings around rounded boulders of country-rock, is even more striking in appearance than that of the Bassick mine.

Our studies of this deposit, comprising two or three visits made in different years when the mine happened to be in active

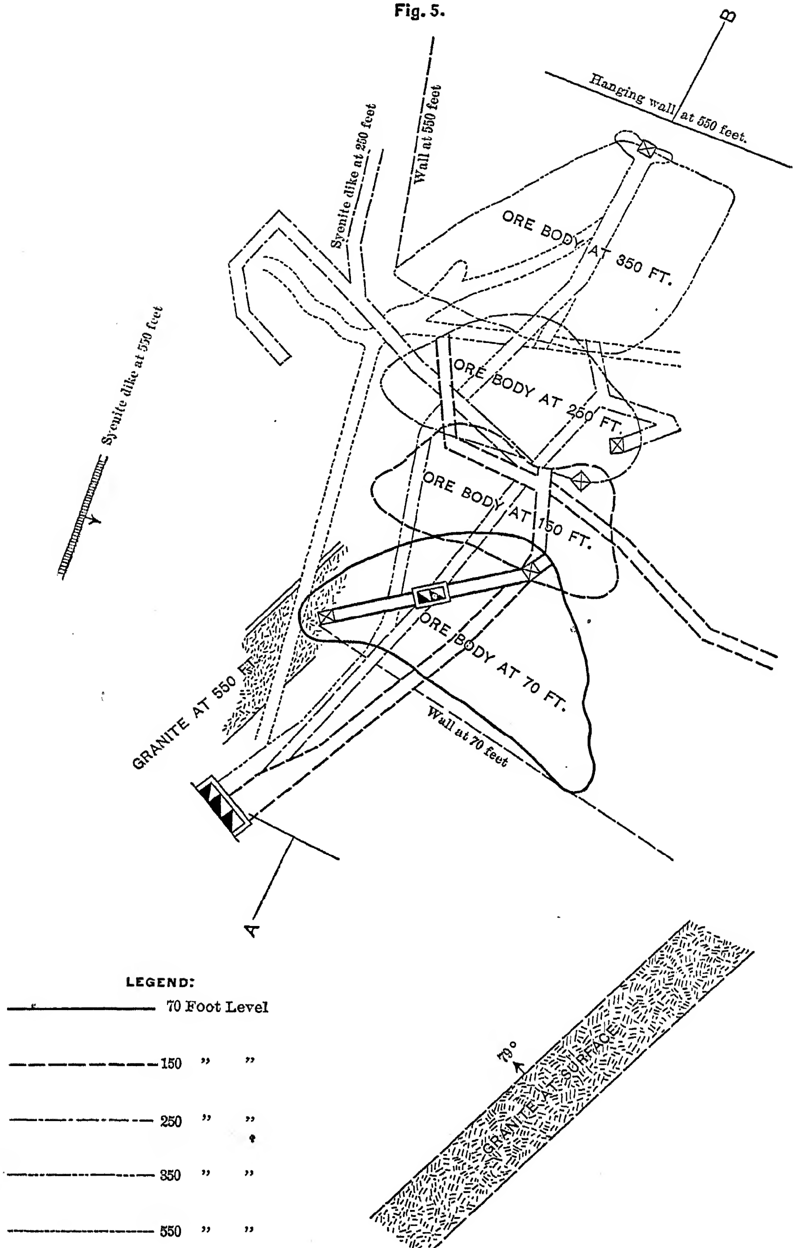
operation, have been somewhat less unsatisfactory than those of the Bassick mine; yet our examination has been necessarily incomplete, as the workings soon become inaccessible after the ore has been extracted, owing to the crumbling of most of the country-rock.

A general idea of the form and character of the deposit will be best obtained by reference to the plan and section in Figs. 5 and 6, respectively. Fig. 5 gives all the reliable data it has been possible to obtain with regard to the phenomena on the principal levels of the mine, projected on a horizontal plane. The outlines of the ore-body there given are those furnished by the mine-surveyor, as it was impossible to obtain them by personal observation. They indicate, therefore, rather the limit of pay-ore than the actual geological boundary of the area of ore-deposition, but are necessarily within the limits of the latter.

In the section on line AB, Fig. 6, is given the form of the pay-ore body, as constructed from similar data. It has been impossible, however, to determine accurately the form of the conglomerate-body which incloses this ore-body, and contains more or less ore, disseminated in scattered grains and crystals.

Mineralogical Character of the Ore.—The boulders or rock-fragments which, though they carry no metallic minerals within their mass, constitute the largest and most striking part of the ore-body, are pieces of the adjoining country-rock (gneiss, granite, or syenite), which were all evidently once angular, but which, through some agency, have become more or less rounded. The gneiss, as might be expected from its softer and more easily decomposable character, is more completely rounded than the other rocks and is generally pretty thoroughly decomposed; but even this has sometimes angular outlines on one side. The granite is also pretty well rounded, as a rule, but is less decomposed than the gneissic boulders. Both gneiss and granite are rich in quartz, and the latter generally resembles the rock of the dike-mass of pink granite which outcrops just east of the shaft-house. The syenite, on the other hand, which is exactly like the rock of the dike (red, fine-grained, and without quartz), is almost invariably angular, only the sharper edges being slightly rounded and the rock entirely unaltered. The boulders vary

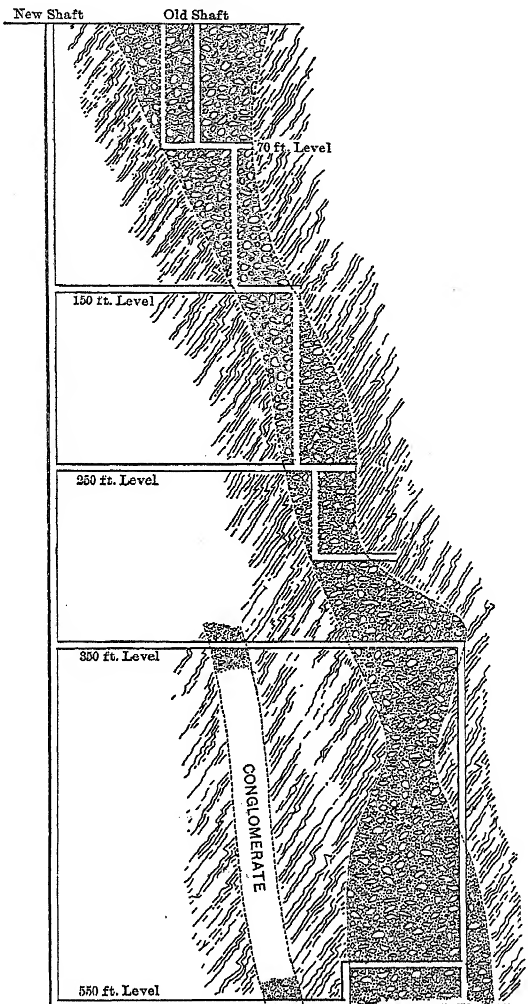
Fig. 5.



HORIZONTAL PROJECTION OF BULL-DOMINGO MINE-WORKINGS.

in size very greatly; some are several feet in diameter, but, as a rule, they are from 1 foot down to 2 or 3 inches. The matrix

Fig. 6.



SECTION THROUGH BULL-DOMINGO MINE-WORKINGS.

between the boulders is of the same material as the boulders themselves, but is very fine-grained. It is mostly decomposed and disintegrated gneiss, with occasional pebbles and grains of granite and syenite.

The minerals deposited around these boulders are usually few in number, comprising galena (generally well crystallized); zinc-blende (of the dark ferriferous variety, and usually in fibrous rather than crystalline form); pyrite (disseminated in very limited amount through portions of the body); and as earthy or gangue-minerals, calcite, dolomite, ankerite or siderite, and quartz (usually in the form of chalcedony). The shell formed of some or all of these minerals is generally from a half-inch to an inch thick. As a rule, it completely surrounds the boulder; in other words, there is usually a double band separating two adjacent boulders. But sometimes the bands are wanting, and the boulders are in actual contact. Three or more adjoining boulders usually enclose a small open vug-like space not filled by the bands of vein-material.

The order of deposition of these minerals appears to have been as follows: The first band around the boulder is either galena alone, or more generally galena and zinc-blende interchangeably, and separated by no sharp dividing line; the galena, however, being crystalline, and the zinc-blende amorphous or fibrous in structure. Upon this band is sometimes a slight coating of dull, lusterless galena in octahedral crystals. A few specks of pyrite are sometimes associated with these minerals. Outside the metalliferous bands comes the various spars, which always form the interior lining of vugs. The order of deposition of these, as observed in some of the larger vugs, is (1) white dolomite; (2) ankerite or siderite; (3) light-colored calcite; (4) white or yellow chalcedony in botryoidal form.

The silver seems to be mostly in the galena. The coarse-grained galena runs 79 to 82 per cent. in lead, and carries about 68 ounces of silver per ton. The fine-grained galena, on the other hand, is considered to contain, on the average, less of either metal.

Form of the Ore-Body.—From a geological standpoint, the outlines of the zone or channel of conglomerate or breccia, the impregnation of which constitutes the ore-body, are more important than those of the ore-body itself. But as it was only in the pay-mineral that the owners of the mine were interested, they paid little attention to the extent or form of the barren boulder-mass which surrounded it.

It was, however, found as a matter of practical experience,

that there was no sharp line of division between pay-ore and barren boulder-mass, but that the one graduated insensibly into the other. On the other hand, the ore-body generally had one, sometimes two or more, sharp boundary-lines, evidently planes of fracture, which were often accompanied by a change of rock on the other side to granite or syenite, as the case might be. Granite was generally found on the hanging- or northeast wall of the ore-channel. Syenite, when it was found, was on the west wall, and had nearly the strike of the syenite dike at the surface. Thus, on the 150-foot level, the ore-body appeared to have a rudely elliptical shape, 90 by 40 feet in dimensions, with a wall of granite on the northeast; it passed into solid gneiss on the northwest, while on the southeast side it graduated everywhere into a barren conglomerate.

On the 250-foot level the syenite dike was observed on the northwest side of the ore-body, though apparently not in immediate contact with the pay-ore. No granite was here detected on the hanging or northeast wall, but a wall was found on the east, running north and south with nearly vertical dip.

At the 550-foot level, the ore-body lies about 150 feet north of the shaft. On this level the miners thought they had two ore-bodies, one east of the other, and each having a longer axis north and south, with a barren zone of boulder-conglomerate between them. Both ore-bodies were surrounded by barren conglomerate, except on the north, where there seemed to be a pretty well-defined hanging-wall. On the west the space between the boulders was largely occupied by calcite and other spars, but carried no ore. There was the appearance on this side of a wall running north and south, but exploration had not been carried systematically to the limits of the boulder-zone, so that it was impossible to define its shape; and it can only be said that its greatest extent apparently lies in a northwest-southeast direction, as above, and that it appears to have grown larger in depth.

At the angle of the drift connecting the shaft with the ore-body a drift to the northwest, just before the ore-body is reached, follows a narrow zone of what appears to be a friction-breccia, where both walls and rock-fragments are highly decomposed gneiss; the inclosed fragments are rounded, as if by the disintegrating action of percolating waters, for the matrix

is the same material as the boulders, only a little more disintegrated, and seems to curve around the boulders, as if it had gradually flaked off. This boulder-zone is like that in which the ore-body occurs, except that the average size of the boulders is much smaller, and no granite or syenite fragments were found in it. It is about 10 feet wide, and was followed a considerable distance, the exact length of the drift being unknown, as it had caved in. A similar boulder-zone, said to have been cut in the 350-foot level, could not be examined. It may be on the same fracture-plane or on a parallel one. The dike of pink granite was also cut in the 550-foot level, a short distance from the shaft, and crossing the drift at an acute angle, as shown on the plan, Fig. 5.

In a general way the phenomena seem to be on a larger scale at this level than on those above. The boulders are larger on the average, the scales of minerals thicker; the galena generally of larger grain and in greater proportion as compared with the zinc-blende, and the vug-like spaces between the boulders larger and more frequent.

No fragments of recent eruptive rock were observed among the boulders in the mine or ore-bins; and, indeed, with the exception of a single pebble of quartz-porphyry (which carried no mineral scales), no rock other than gneiss, granite, or syenite was observed in the mine at all.

With regard to work done in the mine since the summer of 1890, the writer has been unable to obtain any information. It is his impression that the mine was closed down soon after his visit at that time.

Genesis of the Ore.—The facts above enumerated with regard to this ore-deposit, though unfortunately meager and incomplete, are sufficient to show a certain resemblance between it and that of the Bassick mine. It is quite possible, however, that if it had been practicable to study both deposits more extensively, the points of difference would have been found to be more numerous or more pronounced. The striking points of resemblance are, of course, that the ore occupies a nearly vertical, chimney-like channel in a conglomerate or breccia-mass; and that the minerals are deposited in concentric scales around the boulders or rock-fragments. The points of contrast are: (1) that the Bassick ore-body occurs not only in the midst of

recent igneous rocks, but in what appears to have been the actual vent or channel through which an explosive eruption took place, while the Bull-Domingo ore-body is entirely in very ancient rocks, and the nearest known recent igneous rocks are almost a mile, and their probable vent twice as far, away; (2) that, while the Bassick deposits are almost exclusively of minerals that might have been deposited from gaseous solutions, those of the Bull-Domingo are pronouncedly such as must have been formed by aqueous deposition. Moreover, the exposed evidences of fracturing and faulting are much more frequent and pronounced in the Bull-Domingo than in the Bassick mine; and although this may result in part from the better opportunities for personal examination afforded in the former case, the fact that walls of solid rock are found within limited distances from its ore-body, and the general elongated form indicated for the boulder- or conglomerate-zone or chimney, are valid indications of a difference in origin and manner of formation from the Bassick agglomerate. To arrive at a completely satisfactory conclusion in regard to the genesis of the Bull-Domingo boulder-zone, it would be necessary to have access to a much larger portion of the country-rock surrounding the ore-shoots than would have been possible, even had all the existing drifts been open to observation.

In the present state of his knowledge, therefore, the conclusion of the writer is that, although craters of explosion are known to exist in which the materials thrown out have in part fallen back and filled up the orifice from which they were ejected (such, for instance, as the Maare of the Eifel, or, to quote an instance nearer at hand, the Coon Butte in Arizona), the Bull-Domingo ore-channel can hardly be considered to have been such a crater of explosion, but was primarily formed by the complicated intersection of a number of fracture-planes, which produced a zone of broken country-rock, in which the included fragments may have been somewhat rounded by attrition, but were, more probably, completely rounded by the solvent action of percolating solutions. It seems quite possible that, at the same time, explosive eruptions of igneous rocks were taking place in the adjoining regions; and by the force of such explosions heated gases or waters may have been injected through the fissures of the surrounding country-rocks, and,

passing through this broken zone, may have rendered more complete, partly by attrition and partly by solvent action, the rounding of the rock-fragments in this zone, and in that case probably may have ejected some of them from near the then existing surface. As it is probable, however, that several hundred feet of rock-material have been eroded off the surface since that time, it is not conceivable to the writer that such fragments could have fallen back freely into such a narrow and intricate channel as this appears to have been. At the best, such a hypothesis seems at present purely conjectural, and not directly proved by known facts.

The deposition of the ore seems distinctly to have been made by aqueous solutions, and not to differ essentially, except in the form of the ore-channel, and the character of its previous filling, from that of the ordinary vein-deposit. More or less rounded fragments of country-rocks coated with concentric shells of metallic and other vein-minerals are known to occur in well-defined fissure-veins. Instances are noted in the present paper in the case of the Humboldt vein. But in such cases there is no evidence that the manner of deposition of the mineral has been different from that in other parts of the vein.

In the case of the Bull-Domingo mine, as has already been said with regard to the Bassick ore-body, it is, in the opinion of the writer, quite within the bounds of possibility, and even of probability, that further explorations beyond the line of the present ore-chimney, along lines of fracturing and faulting, may disclose other valuable ore-bodies, the existence of which is not suspected at present.

Mines in Rhyolite near Silver Cliff.

Geological Sketch.—The rhyolite area near Silver Cliff includes what may be called the Silver Cliff plateau, with Round mountain and the intervening valley. The plateau is about 2 miles long and 1 mile wide. From its northern part rise the White hills, which have no special topographic importance, as their highest point is only 400 feet above the northern edge of the rhyolite mass. Round mountain, on the other hand, is a quite sharply pointed conical hill, so steep-sided as to constitute an important topographic feature, although its elevation above the plains around it is barely 700 feet. The summit of Round

mountain is dense banded rhyolite, with steep, irregular dip; the southern end is breccia containing fragments of the banded rock. There are slight exposures of glassy forms of rhyolite on the lower slopes. On the east, the Archæan rocks extend half-way up the side of the mountain, and the contact between them and the rhyolite is vertical, or dips steeply to the west. This mountain is supposed to be at the vent from which the rhyolite of the plateau was poured out.

The Silver Cliff plateau occupies the site of a former basin, in which at one time there was probably a lake. At the time of the rhyolitic outburst of the Rosita hills there was a local eruption of the same character in this region, commencing with showers of volcanic ash and of rock-fragments, which filled the lake and built up about it hills which have since been removed in great measure by erosion. At present the southern half of the plateau is capped by solid lava to a depth in places of 150 feet. The cliff of blackened rhyolite on the southern edge, where the main discovery of ore was made, is 30 to 50 feet high. In many places, as in the Vanderbilt mine, the rock is plainly fragmental and stratified, and has a well-defined dip. The contact of Archæan and rhyolite along the western border is a gently undulating surface, and in most of the prospect-holes the Archæan is much broken and resembles a breccia. The northern half of the area is breccia and tuff, except a few dikes of massive rock. At the Songbird and Mountain View mines, and along the western border generally, the gneiss under the rhyolite has been much altered and is locally ore-bearing, carrying magnetite, pyrite, and some galena, as in the Immortal and Keystone mines. Near the Sunrise, and along the eastern border as far south as the Vanderbilt, the rock is a finely-bedded tuff, dipping south and west. The thickness of fragmental material below the highest point of the hills is more than 550 feet. These beds terminate abruptly to the south along an east and west line running near the Vanderbilt, which Mr. Cross thinks may be a fault-line.

The massive rock is everywhere characterized by a banded or fluidal structure, and in it topaz and garnet have been found. Under the massive lava on the southern portion of the plateau is pitchstone or glassy rhyolite, about 50 feet thick, with about as much more below, containing spherulites, which, when de-

composed, form a boulder-zone. These glassy rocks outcrop around the cliff to the south and east, and are found in cellars in the town of Silver Cliff.

Surface-Deposits.—The original outcrop of the ore-bearing rhyolite on the Silver Cliff and Racine Boy claims was apparently nothing more than the ordinary banded rhyolite, stained and blackened by oxides of manganese, extensively cracked and fissured, and carrying little flakes of chloride of silver in the cracks. As far as known, no other metallic minerals were detected, nor was there any definite boundary or regularity of form to the part that constituted the ore. An area several hundred feet in diameter and 30 to 50 feet thick was thus found to be ore-bearing. When examined by us in the quarry, the principal set of joints or rock fractures were observed to run nearly northwest and southeast, and it was on these that the most silver was found. On some of these cracks was a considerable coating of clear black manganese oxide; in others, where there was more iron oxide, the coating had a metallic luster; and it was on the latter, according to the observations of the miners who were sorting the ore, that the principal values were found. A set of secondary joints or fractures, crossing the main joints nearly at right angles and reaching to the surface, could be observed along the benches of the quarry. These also, were heavily coated with manganese oxide, and carried ore. It was but rarely, at that time, that the flakes of horn-silver could be detected by the naked eye. Our observations indicated that the horn-silver was more frequently deposited on small cracks, adjoining those filled by iron and manganese oxides, and apparently of later formation. The light-colored mass of the rock had a faint pink tinge, and a specimen analyzed contained 0.06 per cent. of manganese oxide. It was the experience of the miners that the silver-values did not occur outside of the stained zone.

When the ore-body was first worked, it is said to have contained from 35 to 50 ounces of silver per ton, but it gradually decreased in value as it was taken at a greater distance from the surface. It is said that, while the mills were running, the rock was not sorted, but sent in bulk to the crusher. The last mill-runs are said to have assayed only about 7 ounces to the ton, and the greater part of this went off in the tailings.

The ore taken from the quarry was sorted, so as to average 50 to 60 ounces per ton at one time; but this fell off later, and it was apparently so low finally as not to pay for working.

It has been a cause of much fruitless speculation that the amalgamating mills were so unsuccessful in treating this ore. It is generally conceded that much the larger portion of the silver was carried away in the tailings, which were afterwards profitably concentrated by hand-jigs. A sample of these tailings, carefully quartered down, yielded in the laboratory of the survey 0.13 per cent. of sulphur, which is sufficient to combine with the silver contained and form sulphides. It is also said that a small amount of antimony has been found in the ore by those who smelted it.

If the silver is generally in the form of sulphide, it would naturally be difficult of amalgamation, and the presence of antimony would heighten that difficulty.

Small amounts of ore were also found near the surface at many other points on the plateau, which, though not comparable in amount to the Silver Cliff body, were sufficient to encourage prospecting to such an extent that over 400 prospect-holes were counted there at the time of our examination. For the most part they had been already abandoned, and there was nothing to show how the ore, if any there was, occurred. Among the more prominent ones, which actually produced considerable values, may be named the Boulder, Vanderbilt, King of the Valley and Silver Bar (formerly the Kate).

Deep Deposits of the Geyser Mine.

The only mine-workings that have extended to any considerable depth on the plateau, say over 100 feet, are those of the Security-Geyser mine. As far as is known, the ore of all the plateau-deposits had given out, really or apparently, within considerably less than 100 feet of the surface. The ore was always chloride of silver where its character could be distinguished. That from the Kate (Silver Bar) claim, worked in early days, is said to have contained some gold also; but this is the only case reported and the statement has not been verified. It does not seem likely that silver would be accompanied by gold in one place and free from it in all the others. As will be seen later, of the two shipments to smelters of ore from the bot-

tom of the Geyser shaft, one contained only one-tenth of an ounce of gold per ton, the other but a trace.

It is only through the underground workings of the Geyser shaft, therefore, that it has been possible to obtain any information with regard to the conditions of ore-deposition in depth. The data which it has been possible to obtain with regard to them in occasional visits during past years will therefore be given in considerable detail.

The Geyser shaft, as it is now called, is located 350 feet north, a little west, of the mouth of the adit leading from the floor of the Silver Cliff quarry, and its collar is 104 feet above that level. It was originally intended to sink the shaft only 500 feet, it being supposed, from the position of the various observable contacts of the rhyolite with the underlying Archæan, that the former was a rather shallow body, and that the underlying granite and gneiss would be reached within the depth named. It was, later, decided to prepare for greater depths, and the hoisting-machinery was given greater capacity. The limit of this capacity was reached in the summer of 1894, when a depth of 2100 feet was attained. Entirely new and heavier machinery, with a capacity of 4500 feet, was then ordered, which is now (May, 1896) in working-order, and sinking has been resumed.

Mine-Levels.—The first exploring-levels or drifts were started at a depth of 500 feet. These were run 500 feet west and 700 feet east; likewise some distance in a southerly direction. At 750 feet levels were run to the east and south, and one branch passed directly under the quarry. Below this, levels were run at 1450 feet, 1850 feet, 2000 feet and 2100 feet from the surface, respectively. The general direction of exploration in these levels appears to have been to the west and northwest, but the 1450-foot level had a drift running southward. Accurate maps of the respective levels could not be obtained, but, in a general way, it is estimated that the main exploring-drifts have a linear extent of about 1½ miles at the different levels, and that about 600,000 square feet of area were more or less thoroughly explored. The section in Fig. 7 gives a somewhat diagrammatic representation of the ground explored.

Country-Rocks.—For the first 150 feet the shaft passed through banded rhyolite. In the tunnel leading to the shaft, a narrow

zone or band of this rock was found to be changed into a plastic white clay, which was almost pure kaolin.

Below the solid rhyolite was about 50 feet of pitchstone, then about the same thickness of the boulder or spherulitic zone. Veins and crystals of calcite were found in the rhyolite under the pitchstone. From 250 feet down to about 1900 feet the shaft passed through white rhyolitic tuff and breccia, the former often distinctly stratified and generally looking like white sandstone, much kaolinized. The breccia varies from fine to coarse, and contains fragments of all the varieties of Archæan rocks found in the region; but no eruptive, other than rhyolite, was observed among the fragments. Some of the Archæan fragments are kaolinized and disintegrated; others are quite fresh. The green decomposition-product of hornblende and mica was in one place thought to be a copper-stain.

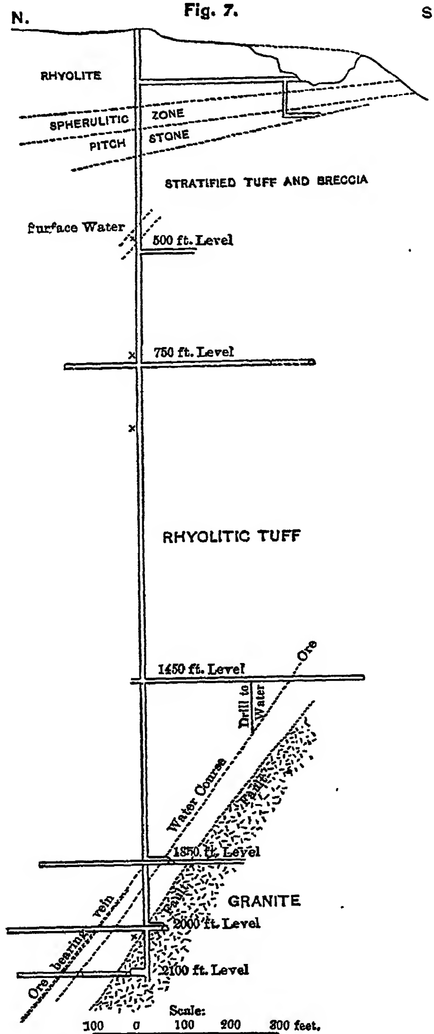
Here and there through the tuff, as far as the 2000-foot level, fragments of charcoal or carbonized wood were observed (the special localities are marked by a cross on the section in Fig. 7). At 335 feet, pieces 2 feet long are said to have been found in the shaft.

The bedding of the tuff was found to be, for the most part, nearly horizontal. In the shaft, a slight dip to the west was noted at times for considerable vertical distances. In the drifts, the dip to the west or northwest is more marked, in a general way, to the southeast of the shaft, and in one place, for a short distance, this dip was 75°. In the shaft there appears to have been a somewhat irregular alternation of white tuff and breccia. For instance, the former was found continuously from 775 to 905 feet, and from there to 1000 feet the rock was mostly breccia, at times with so many large fragments of Archæan, up to 2½ feet in diameter, that it was thought the solid formation would soon be reached.

The shaft actually entered the granite and gneiss of the Archæan near the 1850-foot level. Where cut in the drifts below, the contact of rhyolitic tuff or breccia and Archæan appears to be on fault-planes. The Archæan consists of mica- and hornblende-schists cut by red granite. These rocks are fractured and sheeted in a direction parallel to the ore-bearing veins in the rhyolite; but the actual contact apparently does not, as might be assumed from the section in Fig. 7, conform in every

respect to these planes. The intersections of the contact by drifts are too few to permit the tracing of the shape of the

Fig. 7.



SECTION THROUGH GEYSER SHAFT.

Archæan wall that incloses the rhyolite. It was noted, however, that the drifts pass out of the Archæan into rhyolitic breccia as they go north-northwest or west from the shaft.

In the 2100-foot level, drifts run only north and west, and

the contact, as contrasted with that in the level above, has an inclination to the northeast. The contact on this level is not sharp and well defined, but rather a broken zone, first of very coarse fragments of granite and gneiss, then of normal rhyolitic breccia, with small fragments of granite and gneiss. The three lower levels have been run 300 to 500 feet north and west in this material, which is sometimes hard and jaspery and of dark red color, but bleaches on exposure to the air. It is traversed by planes of movement, sometimes irregular and curving, but all having a general northwest strike. Beyond one of these planes is a dark bluish rock, supposed by the miners to be limestone because it effervesces freely with acid, but, on microscopic examination, found to be a decomposed basic eruptive, containing considerable calcite.

Ore-Bodies.—No defined ore-body was found until the 1850-foot level was reached. Thin films or stains of metallic sulphides, said to assay high in silver, occurred occasionally in the shaft and in some of the drifts, lining delicate cracks in the tuff, and sometimes also in the Archæan fragments. On the 1450-foot level, about 450 feet south of the shaft, it is said that there was found in the white tuff a quarter-inch seam of ruby silver and argentite, with crystalline calcite, which had a general course north-northwest, and was traced about 50 feet, when it disappeared.

The main ore-vein was first found in the 1850-foot level, about 200 feet northwest of the shaft, as a narrow seam, a fraction of an inch wide, with barite and calcite gangue, which widened, as it was followed, to 4 or 5 inches, mostly of galena, and narrowed again to a mere knife-edge seam in about 150 feet. A few nearly parallel seams, containing only calcite and barite, were observed near it. This vein was traced by a winze downward, and was cut afterwards in the 2000-foot and 2100-foot levels, gaining width and richness as it went down. In the 2000-foot level, it is quite thin in the middle, and splits into several thin seams at the northwest, which gradually wedge out.. On the 2100-foot level a smaller vein is found about 40 feet northeast of and parallel to the main vein; this also splits at the northwest end. The general strike of the vein is N. 37° to 40° W., and it stands nearly vertical, its average dip from the 1850-foot to the 2100-foot level being 70° to the northeast.

Vein-Materials.—Although very thin, in no case attaining as much as a foot in width, this vein has been remarkably productive, owing to the richness of the ore and the relatively small proportion of gangue. The principal metallic minerals are galena, zinc-blende, chalcopyrite, cupriferous argentite, tetrahedrite, ruby silver, and possibly stromeyerite or polybasite. Hessite and leaf-gold are said to occur, but their presence could not be verified. The galena occurs in fine scaly form, rather than in the usual massive crystals. The zinc-blende is generally of the ferriferous variety known as "black jack" by the miners, and is rather porous. It occurs at times in cup-shaped forms, which are lined with fine crystals. It sometimes forms cross-courses, or distinct shoots in the vein, which carry from 300 to 400 ounces of silver to the ton, and always a good deal of galena. The chalcopyrite occurs mostly amorphous and easily disintegrable, in rounded patches, distinct from the other minerals, and prominent by its dull brass-yellow color. Where the ore occurs in botryoidal form, one can distinguish the following succession from the center outward: (1) barite in tabular crystals; (2) galena (and argentite); (3) copper sulphide; (4) gray copper in crystals; (5) small crystals of chalcopyrite. In certain parts of the vein which consist exclusively of metallic minerals, they have a peculiarly fresh look, as if quite recently deposited and not yet completely consolidated. The average composition of the ore is best shown by the following analyses of two car-load lots kindly furnished by the Arkansas Valley Smelting Company, of Leadville, to which they had been sold:

Ore from Geyser Mine.

	Lot No. 1.	Lot No. 2.
Gold	Trace.	(*a*)
Silver	*b* 1.05	*c* 1.27
Lead	23.80	17.60
Zinc	14.00	11.10
Copper (wet assay)	1.50	2.30
Iron	2.30	2.00
Manganese	1.20	.80
Lime	1.70	......
Sulphur	12.60	9.50
Silica	33.60	46.90
Total	91.75	91.47

a 0.10 ounce per ton. *b* 250.42 ounces. *c* 300.28 ounces.

In addition to the above metals, there was probably antimony, which had been proved qualitatively, in the mineral that was supposed at the mine to be hessite, but is probably either tetrahedrite or polybasite. Barium and alumina probably make up a part of the balance.

Of earthy minerals, the most common are barite, calcite and quartz, the latter generally in the chalcedonic form. Fragments of country-rock are found in the vein, more or less rounded, and changed on the outer part into hornstone-like material, which, in turn, is coated with galena, barite, etc. Rounded cavities in the vein material are often filled with a white powder, apparently an infiltrated decomposition-product of the rhyolitic tuff.

Water-Courses.—This mine has proved unusually dry for the region. In the upper part of the shaft the first considerable flows of water came in at 340, 390 and especially at 420 feet. This water, collected at the 500-foot level, amounted to 250 gallons per minute, and was undoubtedly vadose or surface-water, and probably seeped in from the surrounding country. At 945 feet it had decreased to 65 gallons per minute, and below 1000 feet had practically ceased, the little water that was found being probably due to leakage along the shaft.

In the 1450-foot level what may be considered subterranean or deep waters were first struck. They were not very abundant, and were only slightly charged with gas. At one point on the south drift there was a slight deposit of tufa. A considerable flow was obtained from a vertical drill-hole sunk 300 feet downward from this level.

In the 1850-, 2000-, and 2100-foot levels there are many small water-courses, from which proceeds a constant flow of water, not very great in aggregate amount, but highly charged with carbonic acid gas, so that there is a constant hissing, sputtering and rumbling, and the water is ejected with such force as to go entirely across the drift at some points. These waters apparently ascend along fissures having a general parallelism with the ore-bearing fissure; but no water proceeds from the vein itself. They often come into the drifts through small cracks or cross-fissures at an angle with the direction of the main system. As they emerge into the air of the drift they deposit freely a

calcareous tufa or sinter on the wall around the crack or orifice out of which they flow. This sinter is sometimes white, sometimes highly iron-stained; it has the texture and the peculiar wavy or ripple-marked surface characteristic of the sinters of the Yellowstone Park. In some places it takes a pisolitic form. Again its surface has a shiny glaze. It deposits very rapidly in some places. At one point on the 2000-foot level, which had been opened only four months, the water issuing from a minute vertical crack on the side of the drift had built out a little ridge of sinter over the crack, 1½ inches from its base and less than half an inch thick.

The water-courses are most active and abundant near the vein or on the line of its extension. On the 2000-foot level, where the vein splits to the northwest, the water comes in on all sides; and when the shaft was first opened, the escape of carbonic acid gas was so abundant at this point that it filled the lower 5 feet of all the drifts on this level and the shaft below the level so that no light would burn, and the miners were obliged to abandon work until a blower could be put in operation to drive the gas out. Even now, a light is soon extinguished if put at the bottom of the drift. The water-bearing fissures are mostly in the rhyolitic tuff, but a few are found in the Archæan, which shows evidence of faulting within itself, in slickensided clay seams and zones of brecciation. The water-bearing fissures decrease in number and in strength of flow as distance from the line of the ore-bearing fissure increases. The abundant escape of gas is the most striking feature of these water-courses. Even where no water comes into the drift, one can often hear the bubbling and sputtering of the escaping gas in an adjoining fissure.

The temperature could not be accurately determined, but is about the same as that of the air in the drift at the 2000-foot level, viz., 80° F.

Analyses of Sinters.—Three typical specimens of sinter from the 2000-foot level were selected for analysis: one of the perfectly white, with very slight iron-stain; one white and brown (both showing the ripple-marked structure well); a third of the pisolitic sinter, strongly iron-stained. They were analyzed by Mr. W. F. Hillebrand, with the following results:

Analyses of Sinters from the 2000-Foot Level, Geyser Mine.

	White.	White and Brown.	Pisolitic Brown.
Silica and insoluble	0.08	0.10	0.17
CaO	53.11	52.60	52.59
CO_2	42.98	42.57	42.03
Fe_2O_3	.20	1.08	1.82
Mn_2O_3 (Mn_3O_4?)	*a* .026	*a* .03	Undet.
SrO	.17	.26	.22
MgO	1.50	1.39	1.01
K_2O	.03	.03	.04
Na_2O	.17	.16	.09
Li_2O	Trace.	Trace.	Trace.
H_2O below 110° C.	.33	.51	.53
H_2O above 110° C.	.88	.72	.87
SO_3	.29	.50	.58
P_2O_5	Trace.	Trace.	Trace.
Cl	Faint trace.	Faint trace.	Faint trace.
Total	99.766	99.95	99.95

a Manganese was estimated on 34 grammes. The same samples showed also minute traces of lead, copper, nickel, cobalt, zinc, alumina and a double trace of antimony.

On comparing these analyses with those of the waters which follow, it appears that, under the influence of free access of air, with presumably reduced pressure and temperature, the precipitation has been mainly of carbonates of lime, iron and manganese; of the alkalis, magnesia, and of other metals a relatively small proportion seemed to have been precipitated.

Analyses of Waters.—Carboys of water from the 500-foot level, *i.e.*, surface or vadose waters, and of waters from the 2000-foot level were collected with great care by the foreman of the mine under the direction of Mr. C. H. Johnson and sent to Washington for analysis. Upon their arrival, there was found to be considerable sediment in each of the carboys, which had presumably been precipitated during the journey, since they had been filtered through cotton cloth when gathered. Mr. Hillebrand found evidence, however, that the filtering had not been complete, as some splinters of wood were found in the sediment, which casts some doubt on the analysis of the sediment.

In the 42.6 liters of vadose water of which the analysis is given below, there was a deep blackish-brown sediment, containing, however, no organic matter, which, after drying at 110° C., weighed 0.5592 gramme, and gave:

	Grammes.
Ignition, ...	.0642
HCl extract, ...	.1588
Insoluble silica and silicates, ...	.3362
Total, ...	.5592

It was assumed that the silica and silicates must have been mechanically introduced, through want of sufficient precautions in filtering at the mine; and this portion of the sediment was not analyzed. Of the sediment in the carboy of deep water, however, the insoluble portion was analyzed, with the result given in the following tables. The total sediment in the carboy of deep water was 10.6602 grammes, of which 5.0134 grammes were insoluble in dilute HCl (at 110° C.). The filtered vadose water had a slightly alkaline reaction and contained no organic matter. The results of analysis are given in the following tables:

Vadose Water from the Geyser Mine, 500-*Foot Level.*
(W. F. Hillebrand, analyst.)

In Sediment. Soluble in HCl.				In Filtered Water.		Assumed Composition Before Sediment was Deposited.
	Amounts Found. Grammes.		Referred to Parts in 1,000,000 of Water.		Parts in 1,000,000.	
SiO_2	.0012	SiO_2	Trace.	Cl	7.9	7.9
PbO	.0010	Pb	Trace.	SO_4	43.2	43.2
CuO	.0008	Cu	Trace.	CO_3 *a*	108.3	110.5
Fe_2O_3	.0456	Fe	.7	K	10.6	10.6
Al_2O_3, P_2O_5	.0365	Al_2O_3, P_2O_5	.8	Na	36.4	36.4
				Li	Trace.	Trace.
Mn_3O_4 *b*	.0498	Mn	.8	Ca	37.3	37.4
ZnO	.0104	Zn	.2	Mg	12.2	12.25
CaO	.0096	Ca	.1	Pb	....	Trace.
MgO	.0039	Mg	.05	Cu	....	Trace.
CO_2	(*c*)	CO_3 *d*	2.2	Mn	....	0.8
	.1588		4.85	Zn	....	0.2
				Fe	....	0.7
				Al_2O_3, P_2O_5	....	0.8
				SiO_2	25.9	25.9
					281.8	286.65
				Free and semi-combined CO_2,	38.8	37.2
					320.6	323.85
				Total CO_2,	118.2	

a Calculated. *b* Assumed condition.
c Any traces of CO_2 present have been neglected.
d Calculated for the metals, as carbonates before deposition.
e No tests for other possible constituents were made.

Deep Water from the Geyser Mine, 2000-*Foot Level.*

Specific Gravity 27° C. = 1.0036. (W. F. Hillebrand, analyst.)

Composition of Sediment. (From 43.76 Kilogrammes of Water.) Insoluble in Dilute HCl. Chiefly Clayey and Feldspathic Matter. Weight Dried at 110° C. 5.0134 Grammes.		HCl Extract. Amounts Found.	HCl Extract. Referred to Parts in 1,000,000.		Composition of Filtered Water.	Parts in 1,000,000.	Assumed Composition Before Sediment was Deposited.
					Cl	186.40	186.40
					Br and I	Traces.	Trace.
					SO_4	161.70	161.70
	Per cent.	Grammes.			PO_4	Trace.	Trace.
SiO_2	54.07	.0667	SiO_2	1.52	NO_3 *f*	1.60	1.60
Al_2O_3		*a* .0461	Al_2O_3	1.06	B_4O_7	Trace.	Trace.
Fe_2O_3	29.70	.2100	Fe	3.36	CO_3 *g*	1437.26	1513.44
TiO_2					Fl	None.	Trace. *h*
	A little				K	198.00	198.00
CaO	(as	2.7921	Ca	45.57	Na	719.45	719.45
	$CaFl_2$?)				Si	2.85	2.85
MgO	None.	.0778	Mg	1.07	Ca	100.84	146.41
K_2O	4.40	Not	....	......	Sr	1.10	1.95
Na_2O	.37	tested	....	......	Mg	176.60	177.67
SO_3	....	for.	....	......	Pb	.05	1.35
CO_2	....	2.3153 *b*	CO_3 *c*	76.18	Cu	Trace.	.02
SrO	....	.0440	Sr	.85	Mn	.19	.57
BaO	....	None.	Ba	None.	Zn	.17	.34
Fl	A little.	......	....	......	Fe	.14	3.50
PbO	....	.0610	Pb	1.30	Al_2O_3	None.	*i* 1.06
CuO	....	.0011	Cu	.02	SiO_2	22.90	24.42
Mn_3O_4 *d*	....	.0232	Mn	.38	Org. mat.	Not est.	Not est.
ZnO	....	.0095	Zn	.17			
Ignition	9.47	(*e*)		(*e*)		3009.25	3140.73
	98.01	5.6468		131.48	Total CO_2	2472.60	2528.46

a Perhaps derived from the mechanically included minerals of the sediment. Contains a little P_2O_5.

b The sediment was largely incrusted on the glass of the carboy and could only be removed by acid; hence the CO_2 was calculated for PbO, ZnO, SrO, CaO, MgO, as normal carbonates.

c This value includes the CO_2 needed by Fe, Mn and Cu, as well as the metals named in the preceding note.

d Assumed condition; perhaps partly as $MnCO_3$.

e Organic matter not estimated.

f Approximation, not a maximum.

g Calculated for normal carbonates.

h The fluorine of the insoluble part of the sediment is probably to be credited to the water.

i Possibly from the insoluble sediment.

These tables present a remarkably complete series of actual analyses, representing:

1. The average contents of a vein-deposit of metallic minerals, rich in silver, lead, copper and zinc, which was first found at over 1800 feet below the present surface.

2. The contents of subterranean mine-waters taken at 2000 feet below the surface, and evidently coming from still greater depths, very highly charged with carbonic acid, and carrying small amounts of most of the metals that occur in the deposit; also the sinter deposited by these waters as they issue from the rock into the mine-drifts—that is, under ordinary atmospheric pressure.

3. The contents of atmospheric waters coming from the surface, which, in their downward course, had traversed rocks similar to those in which the first-named deposit is inclosed, and through which it may be assumed that metallic minerals similar to those in the deposit may be sparingly disseminated.

From these analyses it is possible to apply a practical test to some of the assumed theories of ore-deposition.

Source of Solid Constituents in the Waters.—In the first place, in comparing the contents of the vadose and deep waters, it is seen that though the latter contain about twenty times as much dissolved matter as the former, the relative proportions of the principal constituents are sufficiently alike to permit the assumption that they have been derived from a similar source; and this source, in the case of the surface-waters, which should have been practically pure when they entered the rocks, must have been the material of the various rocks through which they have passed in reaching a depth of 500 feet below the surface. In making this comparison, one must bear in mind that the deep waters had already deposited the greater part of their lime as sinter before they were analyzed; hence, while the vadose waters contain three times as much lime as magnesia, in the analysis of the deep waters lime is to magnesia in the proportion of only 4 to 5.

The alkalies appear in the same relative proportions in each case, though the aggregate amount of the two constituents is proportionately less in the vadose than in deep waters.

Iron and manganese are in nearly equal amounts in the vadose waters, the latter slightly predominating, whereas in the deep waters iron is in a hundred-fold greater relative amount. This might be explained by the relatively larger amount of manganese oxides present in the surface-rocks. It has often been noted by the writer that manganese oxides are generally present in much larger proportion in the oxidized portions of ore-bodies than below the zone of oxidation, which may be due to

their forming, in contact with atmospheric agents, less soluble salts than do the iron oxides.

The other metals are in such small proportions in either case that one cannot reason from their relative amounts; and it is not surprising that most of them could not be detected in the vadose waters.

Such constituents as fluorine, boric and nitric acids, strontium and barium are characteristic of deep sources, but they also might have been present in the vadose waters without being detected in the small amount of solid constituents available for analysis.

The greatest apparent discrepancy is the tenfold greater proportion of silica in vadose over deep waters; but this is likely to have arisen from excess of CO_2 in deep waters which would throw down silica in solution, or from the uncertainty in determining what part of the solid material in the waters was due to mechanical admixture, and might hence be neglected.

In the deep waters chlorine and sulphuric acid are present in about equal proportions, and carbonic acid is greatly in excess of both combined. In the vadose waters, while carbonic acid is still in excess, sulphuric acid appears in relatively greater, and chlorine in relatively smaller, proportion than in the deep waters. In what manner they would combine with the several bases in either case it is of course impossible to say definitely.

The deductions which the writer draws from these considerations are:

1. That inasmuch as the surface-waters must evidently have derived the substances they contain in solution from the rocks through which they have passed in seeping downward, it is fair to assume that, in like manner, the deep waters have obtained their constituents (in great part, at any rate) from surrounding rocks, and not necessarily at very great distances from where they now issue, since through higher temperature and greater acid contents they would probably have been more active solvents than the surface-waters.

2. On the other hand, the great excess of carbonic acid in the deep waters, combined with the presence of fluorine, boric acid and chlorine in considerable amount, point to a source where chemical decomposition is actively going on, which might readily be supposed to be a body of still uncooled igneous

rocks, which surface waters had reached, and from which they were sent back toward the surface again along the present lines of fissuring. Although the deep waters contain most of the metals found in the vein-deposit of the Geyser mine, it is not easy to conceive how the metallic sulphides of that deposit could have been derived from the waters of such chemical composition as these; and it seems more reasonable to assume that the vein-minerals were deposited by earlier waters of somewhat different composition, carrying more barium and silica, and characterized by sulphuretted hydrogen rather than by carbonic acid.

3. The conditions here indicated seem to negative the prevalent belief that a decrease of temperature and pressure are the principal determining causes of the precipitation of vein-minerals from ascending solutions. In the earlier deposits abundant precipitation ceased before the marked decrease of temperature and pressure that accompanies an approach to the actual rock-surface was reached; and in the modern mine-openings, where present ascending waters have been artificially cooled and relieved from pressure, the abundant deposit has been, like that of thermal springs at the surface, mainly of carbonate of lime and oxide of iron, and contains only faint traces of the other vein-materials that make up the bulk of the neighboring vein-deposits.

4. It might be assumed that the surface-deposits of chloride of silver and oxides of manganese and iron which are thinly and irregularly disseminated through the rhyolite near the actual surface, were precipitated from the carbonated waters at a time when they reached the present surface, the oxides having been originally carbonates, and the silver chloride having been deposited as such, and that these deposits are therefore a later phase of ore-deposition than the vein-minerals. A certain color of probability is lent to this hypothesis by the fact noted by Mr. Johnson, superintendent of the Geyser mine, that there is evidence of an escape of warm air or gas through holes at the surface, which in cold weather is visible as steam, along a zone about 100 feet wide, running east and west through the Geyser shaft-house. Moreover, fluorite and barite are said to have occurred, associated as gangue-minerals with chloride of silver, in the Silver Bar (formerly Kate) mine.

On the other hand, all our evidence goes to show that the chlorides and oxides pass into sulphides at short distances below the surface, and that here, as in other deposits, the chloride of silver is a secondary alteration by atmospheric agents of an original sulphide. It appears more probable, therefore, that all the metallic minerals of the plateau were formed under the same conditions and during the same general phase of ore-deposition. That they are so irregularly disseminated is probably due to physical rather than to chemical causes. The rhyolitic tuff which forms the main country-rock is so poorly consolidated and of so plastic a nature, that fracture-planes are less continuous and less open in it than in harder and more rigid rocks. Moreover, the natural planes of division, the bedding-planes, are horizontal rather than vertical. Hence there have been no well-defined and continuous water-channels traversing the whole thickness of the mass, but the ascending solutions, after leaving the vicinity of the bounding walls of the harder Archæan rocks, have been obliged to follow devious courses along minute cracks and fissures that were not continuous. Thus, comparatively small amounts of these solutions have reached the upper lavas, and their load has been deposited as thin films in the joints or minute cracks of the rocks. The fact that, at the present time, the descending surface-waters penetrate the mass to so moderate a depth is an argument in favor of this view.

It is probable that the present vein-fissure will soon reach and pass into the Archæan wall-rock, in which it may widen out. It is very uncertain whether, in this case, the ore will continue to be as rich as it has been; for a change in wall-rock is generally accompanied by a change in the character of the ore. These points will soon be settled, however, by actual development, as the workings of the Geyser mine follow the present vein in depth.

General Conclusions.

Forms of the Ore-Bodies.—The preceding pages have been mainly devoted to the description of the four principal mines in the immediate vicinity of Rosita and Silver Cliff. Other important deposits within this area, and in the surrounding region, have not been mentioned, because, owing to the irregu-

lar and disconnected manner in which they have been worked, it has been impossible to obtain any detailed information with regard to them. With but few exceptions, these deposits belong to the type of the Humboldt-Pocahontas vein; that is, they are vein-deposits on fault-planes in some of the many varieties of igneous rocks that outcrop in the region. They are, in general, rather narrow fissures, which do not bear evidence of having at any time constituted large open spaces, but in which the ore-bearing solutions have deposited their contents by first filling the interstices between the sheets of sheared and banded country-rock and afterwards partially replacing these sheets or bands by vein-materials. The ore in these cases is generally confined to the fault-fissure, and the deposits may be characterized as well-defined vein-deposits or true fissure-veins.

The mines of the Silver Cliff plateau show a different type of deposit, but, in the opinion of the writer, the essential differences lie rather in the form of the ore-channels than in the character of the ore-bearing solutions. From Mr. Cross's description of the Democrat and Ben Eaton mines, in the central rhyolitic area, these deposits seem to constitute an intermediate stage between the two types. These mines occur on the south point of Democratic ridge, known as Indian Castle. This is a rounded eruptive channel of rhyolite, in which the rock is massive, brecciated or spherulitic, as the case may be. There are indications that there have been several eruptions. It has since been much altered, and the alteration-products vary from hard quartzite-like material to softer material, resulting from the kaolinization of inter-spherulitic glass. Trachyte dikes run both north and south and east and west through the mountain, and their decomposition-product is usually soft. The ore-bearing fissures run north and south, with a steep eastern dip, through both rhyolite and trachyte. The ore-solutions followed these fissures primarily, but found the softened spherulitic glass and certain brecciated zones also very good channels. The ore is now found in these seams or fissures, but all soft kaolinized parts are likely to be impregnated. The main ore-body was an oval chimney, of varying size, in soft matter, which is connected with a fissure at tunnel-level. From one part, on stoping upward, a soft yellow mud

flowed out, which was found to carry 40 ounces of silver and $14 in gold to the ton. For the most part, the solid masses of ore are less than an inch thick.

It has already been suggested, in the case of the deposits on the Silver Cliff plateau, that the fact that the surface-deposits are not in the form of fissure-veins, as they are found at the bottom of the Geyser shaft, is due to physical causes which have not permitted the formation of long continuous water-channels along fissures. In this case similar irregularities have been produced by chemical causes; but it has been the physical effect—the production of channels of freer flow, through the decomposition of the rock—that has led the ore-depositing currents to leave the regular fissures.

The deposits in the Archæan rocks on the borders of the eruptive region are likewise unusually irregular in form; and, in most of the observed cases, this irregularity may be ascribed to a combination of chemical decomposition with dynamic fracturing of the rocks; that is, while the ore-channels have been primarily determined by the dynamic movements that produce the ordinary rock-fractures, vein-fissures and brecciated zones, on which ore-bodies are generally deposited, their course has been varied, or they have received unusual forms as the result of the energetic dissolving or decomposing action of heated solutions that traversed them during the closing phases of volcanic action in the region. This supposes a prolonged alteration and decomposition of the rocks along the water-channels before the actual deposition of metallic minerals. In some of the observed cases, there are fairly well-defined fissure-veins in the Archæan rocks; but more commonly, in this region, the ore-deposition appears to have taken place along a zone of decomposed rock, which zone was undoubtedly determined in the beginning by dynamic action. The ore-deposition along such zones, as might be expected, has been more irregularly spaced and less concentrated than would have been the case in a fissure which had not been thus enlarged by chemical decomposition. The Bull-Domingo ore-body is apparently an extreme type of such a form.

Whether it be or be not admitted that the boulder-filled channel of the Bull-Domingo represents the neck of an ancient crater of explosion, the Bassick ore-body is unique in the evi-

dence it affords of a direct connection with volcanic agencies; and in the determination of its form dynamic agencies have apparently played a very subordinate part.

Cripple Creek Deposits Compared.—It is interesting to contrast the deposits of this region with those of the now famous Cripple Creek district, which lies in a closely analogous geological position, 40 miles to the northward, and which presents in its geological structure so many points of resemblance. There, as here, the main ore-deposition has taken place in and around a central volcanic focus, where a series of comparatively recent igneous rocks have broken through an older series of pre-Cambrian crystalline rocks. There, as here, the principal deposition has taken place along a system of fracture-planes, traversing both the eruptives and the underlying crystalline complex, and, while not strictly confined to the eruptives, it has been, so far as present developments show, more abundant in the former than in the latter.

In the Cripple Creek region there is one principal and predominant system of mineralized fractures, running about north and south. In this district a system running north and south or a little west of north is apparently the more frequent, but there are also abundant fractures running east and west, and others quartering between the two. The geological history of this region has been more complicated; there have been a greater number of successive eruptions; and it is probably in consequence of this fact that the fracture-systems are more varied and complicated.

Mineralogically the contrast is greater. In Cripple Creek the important metal is gold, deposited mainly in the form of telluride, and the characteristic earthy mineral associated with it is fluorite. Here, gold as telluride occurs in certain parts of the district, and fluorite is sparingly found; but the greater part of the valuable minerals are silver-minerals, in their usual association with sulphides of lead, zinc, and iron, and with barite as the leading gangue-mineral. They differ from the ordinary deposits of this class mainly in their greater average richness.

Source of the Metallic Minerals.—While it is possible, by careful study of the geological and mineralogical conditions of a series of ore-deposits, to find valid reasons why the ore-bearing

solutions deposited their load in certain forms and certain localities, and while reasonable deductions may be made as to the probable direction from which these solutions came, the question as to the source from which the solvents derived the materials which they have thus deposited in the form of ore-bodies is one that trenches somewhat upon the domain of pure speculation. Yet, even here, there are many facts of geological observation that have a distinct bearing, one way or the other, upon the various speculative views that have been put forth by geologists.

The general views of the writer upon this question, as already expressed in earlier publications, are: that the heavy metals have probably been brought up from the interior of the earth within the magmas of igneous rocks, and that by some process of differentiation not yet completely understood, either previous to or during the process of cooling and consolidation, they have been concentrated within certain bodies or parts of bodies of eruptive rocks; and, further, that ore-bodies, as found at the present day, are the result of a concentration (perhaps many times repeated) of the materials thus brought up, which are in all probability very finely disseminated through the present rock-masses or combined in minute amounts in the more common basic minerals. This seems a more rational hypothesis, and one more in accordance with modern scientific practice, than to content oneself with assuming simply that the ascending waters came charged with metallic minerals from the bathysphere, meaning thereby a region in the interior of the earth which is richer in heavy metals than any part of the earth's crust that comes under our observation; for this simple assumption affords no explanation why metallic minerals are concentrated in one part of the earth's crust and not in another, and it supposes a free flow of waters at greater depths than in our present state of knowledge of terrestrial physics it is considered possible that channels which would admit of a flow of water through them could remain open.

Furthermore, the writer's hypothesis admits of a practical test, which is impossible in the other case. If the vein-materials are found to form a constituent part, even in minute traces, of comparatively fresh and unaltered country-rocks in a given ore-bearing region, and at such distances from any water-channels

as to render it improbable that these materials could have been brought in through these channels, it is reasonable to assume that these or similar rocks have been permeated by the waters from which the known ore-deposits were precipitated, and that from them they derived their contained vein-materials. For this reason a series of careful tests of selected country-rock for possible contents in the precious metals was carried on under the direction of the writer at the laboratory of the United States Geological Survey in Denver. Since the office at Denver was broken up, it has not been possible to continue these tests, owing to want of proper facilities in the Washington laboratory.

Such tests of the rocks from this district as were completed (unfortunately very few in number) are given below.

The five assays for silver were made upon four assay-tons of each sample, and blank assays of a like amount of the lead-flux were simultaneously made, the silver-contents of the flux being deducted from that found by the rock-assay.

In the case of the black granite from the Blue mountains, another portion of the rock was pulverized and the constituent minerals were separated by the Sonnstadt solution. The bi-silicates in this case were found, as shown below, to contain both silver and lead; but no silver was found in either quartz or feldspar.

Assays of Custer County Country-Rock for Silver.

(L. G. Eakins, analyst.)

Rock.	Locality.	Silver per Ton.
		Ounces.
Trachyte	600 feet southwest of Humboldt shaft	0.007
Trachyte	Summit of Game Ridge	None.
Rosita breccia	South of Game Ridge	None.
Rosita breccia	South of Game Ridge	None.
Fairview diorite	Mount Fairview	.01
Tyndall andesite	Northeast spur of Mount Tyndall	None.
Rhyolite	Top of Round Mountain	.402
Red granite	Near Haskell's ranch	.005
Black granite	Blue Mountains	.025
Bisilicates of black granite (0.045 per cent. lead)		.04

It thus appears that 5 out of 9 of the rocks tested contain appreciable amounts of silver; and that in one of these rocks both silver and lead were found to be present in combination

with other bases in the bisilicates. It seems, therefore, probable that not only the recent eruptives, but the older granites through which the ascending solutions must have passed, contain enough of the precious metals, and, it may be assumed also, of the other vein-materials to furnish, in the long time that is accorded to the accomplishment of most geological phenomena, sufficient material for the formation of existing ore-bodies. The analysis of the vadose waters in the Geyser mine has demonstrated the capability possessed by even cold surface-waters of taking up such materials in their passage through the rocks. The subterranean waters that were circulating here at the time of the formation of the ore-deposits must have been much more energetic solvents, being heated by contact with the cooling masses of igneous rock, and probably deriving a certain amount of active and energetic mineralizing agents, such as fluorine, chlorine, etc., from these igneous masses at the time of contact. Hence it is fair to assume that the vein-materials in this region were originally derived from both recent and ancient eruptive rocks—a conclusion similar to that arrived at by Mr. Penrose, from his more exhaustive study of the ore-deposits of Cripple Creek.

Biographical Notice of Gabriel Auguste Daubrée.

BY J. F. KEMP, NEW YORK CITY.

(Colorado Meeting, September, 1896.)

THE death, on the 30th of May, 1896, at Paris, of the venerable Gabriel Auguste Daubrée, an honorary member of the Institute, recalls with emphasis the great value of his varied contributions to geology, mineralogy and hydrology. In fact, more and more as the years pass, and as attention is given to the obscure problems of metamorphism, of the development of joints, of slaty cleavage, of schistosity, of fault-systems, of the subterranean circulations of water, and of many other agencies which are often closely involved in the production of ore-bodies, the appreciation of his work increases, and quotations from his papers are made.

Daubrée was born June 25, 1814, at Metz. After receiving

his preliminary education he entered the *École Polytechnique* at Paris, and two years later passed upward into the *École des Mines*, where he was trained as a mining engineer. His preparation for his after-work was thus similar to that of many another geologist, for it may be truly said that since the time of Werner the mining school has been the mother of investigators in this branch of science. On graduation he was attached to the council having supervision of mines, bridges and roads, and in this connection traveled in Great Britain and Germany. Later he was sent with a commission to report on the mineral resources of Algiers, and gained in these trips that breadth of experience and observation so essential to good geological work. In 1838 he was appointed a mining official in the Department of the Upper Rhine, and at the same time Professor of Geology in the Academy of Strasburg. In 1861 his brilliant contributions to geological science gained for him the call to the chair of geology in the *Musée des Sciences Naturelles* at Paris. A year later he was transferred to the chair of geology in the *École des Mines*, and other honors followed one another quickly. In 1867 he became Inspector-General of Mines, and in 1872 Director of the *École des Mines*. In the duties of the last-named post he remained for fourteen years; but with increasing age he resigned them in 1886, and was appointed honorary director in recognition of his distinguished services to science.

Daubrée was one of the principal figures in the synthetical school of French geologists and mineralogists, which has exerted so important an influence upon the scientific thought of the last sixty years, and which has contributed so much in the way of sound and reliable experimental data as foundations for the interpretation of nature. The geological experiments of Sir James Hall, of Edinburgh, in 1812, struck the keynote; but the great development of the theme took place in Paris. Dufrénoy, one of Daubrée's own teachers, was a leader, and many distinguished French mineralogists in earlier and later years have been devoted to the same methods and lines of investigation. In contrast with many of his colleagues in Paris, Daubrée was himself especially attracted to those experiments which led not so much to the formation of individual minerals as to the attainment of geological results.

His first serious essay, that of producing cassiterite, was happily crowned with success, and by it a flood of light was thrown on the geology of tin-deposits. At once the characteristic associates of tin-ores, the fluorine-bearing minerals, and their occurrence around the edges of granite intrusions as a phase of contact-metamorphism, were explained. Happily, too, the experiments gained the attention of his former teachers, Berthier, Élie de Beaumont and Dufrénoy, through whose influence they were presented to the French Academy. After this, Daubrée experimented with a whole series of minerals, in the production of which hydrofluoric and boracic acids played a part. Sulphide ores of various kinds likewise received attention, and old coins and bronze implements from the ancient Roman baths were carefully examined to see what light they could throw on the more obscure questions of the origin of ore-deposits; for, as is shown by the summary of his work, which Daubrée published in 1879,* in these experiments it was the origin of ore-deposits, rather than the production of single minerals, that was always before his mind.

This book itself gives us the best insight into the fields of investigation which received his attention in the years of his prime; and a brief sketch of its contents will set forth the range of his activities. It opens with a description of the experiments to produce the ores and associates of tin and the sulphide-minerals, to which allusion has just been made. After the sulphide-ores, the book takes up the applications of the experimental method to the study of metamorphic and eruptive rocks; and in the course of the preliminary discussion the author introduces the phrase "regional metamorphism," which has met with such wide adoption since. The effects of water, when hot, when in the condition of steam and when superheated, are passed in review, and important observations are recorded from the mineral formations at the hot springs of Plombières and elsewhere. Conclusions regarding the formation of zeolites and siliceous sinters in volcanic rocks, and regarding igneous phenomena and metamorphism, follow. They foreshadow some of the themes treated more in detail in the author's subsequent works on subterranean waters. The ex-

* *Études Synthétiques de Géologie Expérimentale.* A German translation by the author's friend, Dr. Adolf Gurlt, appeared in 1880.

perimental method is next applied to the explanation of volcanic outbreaks and lavas, especially as connected with water.

In the second part of the book the interesting and important experiments in the production and transportation of sediments, and the parallelism shown by the artificial products with those met in nature, are described. These have appealed not alone to geologists, but to engineers as well, and to all engaged in the maintenance of channels and the control of sediments. The experiments to produce joints, faults and all forms of fractures in rocks are, however, the most far-reaching of all in their applications. Daubrée was alive to the importance not only of simple pressures and strains, but likewise of torsional stresses, of whose profound effects upon the crust of the earth we are becoming yearly more thoroughly convinced. And in the recent important investigations of many geologists of today, American as well as foreign, we can trace the influence of these early experiments of the brilliant Frenchman. Nor did the important relations which can be recognized between faults and joints on the one hand and topography on the other fail of elucidation in his researches.

From imitating the phenomena of the earth, the author turns in the closing pages of the book to the reproduction, both chemically and mechanically, of those of meteorites, and, at the same time, casts much light on the problems of our basic and ultra-basic igneous rocks. With the discussion of meteorites the book closes, and it may be truly said that upon all the subjects treated it has left an indelible impress.

In the three large volumes which appeared in 1887, and which represent in part the activity of the eight years following the publication of the work just cited, Daubrée discusses at length the work of water as a geological agent in the past and in the present.* The full titles below will briefly indicate their scope, and make it evident that they elaborate some of the themes outlined in his earlier work. Like that work, they collect and sum up many scattered brochures of the author into

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* The works are: *Les Eaux Souterraines aux Époques anciennes. Rôle qui leur revient dans l'origine et les modifications de la substance de l'écorce terrestre.* Paris. 1887. One large volume.

Les Eaux Souterraines, à l'Époque actuelle, leur régime, leur température, leur composition, au point de vue du rôle qui leur revient dans l'économie de l'écorce terrestre. 1887. Two large volumes.

one connected whole. Ore-deposits and metamorphism are still in the foreground, and were clearly the principal objects which Daubrée had in mind. We need only note the citations of his books in Posepny's recent treatise on ore-deposits, and the still more extended and frequent quotation in all large works on metamorphism, to appreciate his influence upon investigators in both departments.

The qualities of Daubrée's personal character, as exhibited to all who were honored with his friendship, were not less admirable than his intellectual ability and scientific attainments. Those who know him have sustained in his death a loss for which they cannot easily be consoled, and thus feel double reason for sharing the regret expressed by the scientific world at large.

A detailed sketch of Daubrée's life and work, and a bibliography, all by his friend of forty years, Dr. Adolf Gurlt, of Bonn, will be found in the *Zeitschrift für praktische Geologie*, July, 1896, p. 281.

Biographical Notice of J. F. Holloway.

BY JAMES F. LEWIS, CHICAGO, ILL.

(Colorado Meeting, September, 1896.)

THE death of Josephus Flavius Holloway, for twenty-one years an active and beloved member, and twice a Vice-President, of the Institute, cannot be allowed to pass without some expression on the part of his brethren of mingled grief and gratitude—grief that we have lost his genial and inspiring presence; gratitude that we have been privileged to know one who lived

> "To make the world within his reach
> Somewhat the better for his being,
> And gladder for his human speech."

The present sketch of his career and character is offered as a sincere, though inadequate, expression of these feelings.

Mr. Holloway's father, Joseph T. Holloway, born, in 1796, in Sunbury, Pa., and yielding in his youthful prime to the prevailing impulse of his generation, started alone, with knapsack

and long rifle, for what was then "the West." Reaching the Ohio river, and travelling on a flat-boat to its mouth, he penetrated the wilderness of Missouri, encountering, among the Indians who then occupied the territory, much danger and hardship. Returning subsequently to Sunbury, he married there. Perhaps the girl he had left behind him was the cause of his return. Such things have happened; and it has also happened many times, as it did in his case, that the young bride, joining fortunes and fate with the young husband, migrated with him to the mysterious, enticing "West." Joseph Holloway and his wife, loading a one-horse wagon with their household goods, made this adventure, and finally settled in Uniontown, Ohio, where he followed his trade as a cabinet-maker, and where, January 18, 1825, the subject of this sketch was born.

But in 1831, with the mobility characteristic of the pioneer period and its people, they moved again, to the new settlement of Cuyahoga Falls, where, after clearing a plot of ground, Joseph Holloway established a new home, and resumed his business as a cabinet-maker. Here he remained, and in after years was elected justice of the peace of the village. He became also, without any election save that of his own impulse and inspiration, a preacher of the Gospel in all sorts of remote and neglected localities—at country cross-roads, at coal-mines, in school-houses, etc.—never accepting payment for his services, but even carrying with him food and forage for himself and his horse. This self-sacrificing labor, which won for him the title of "Father Holloway," was continued until he died in 1878, at the age of eighty-two. His wife, who passed away in 1863, is described by one who was in youth her nearest neighbor, as "the sweetest woman who ever lived." Yet her sweetness did not prevent her from rearing her children with a strict, though affectionate, discipline.

These particulars of parental character and experience have been stated because they belong to the environment in which the subject of this sketch became a typical American engineer.

Young Holloway was but six years old when his parents moved to Cuyahoga Falls; and in watching "the wheels go round" in the mills run by water-power in that village, he received those first lessons in mechanical engineering which determined his future life.

His sole opportunity for education, furnished by the country-school of that day, must have been enjoyed for a brief period only; for at the age of fourteen he became clerk in a drug store. Fortunately this establishment included a department for the repair of watches and clocks, in which he spent most of his time. At sixteen he was apprenticed as a machinist, and at twenty he went to Massachusetts, where he worked for about a year at his trade. In 1846 he returned to Cuyahoga Falls, and was employed by the Cuyahoga Steam Furnace Company.

During the winter of 1847–48 he designed and built at Cleveland, in connection with Mr. E. H. Reese, the propeller "Niagara," for use upon the lakes. Of the propeller-engine, Mr. Holloway has said:

"It was not much of an engine to look at, but it was a good one to go; and, such as it was, it was built from the drawings made by the boy from the country machine-shop."

When the boat was finished it was loaded with flour, and the two proprietors and builders of the machinery took the craft to Buffalo, where it was to be turned over to the person who had furnished the money to build it.

This trip to Buffalo proved to be the turning-point in his career. While the boat was tied to the dock, awaiting the completion of the delivery, a gentleman stepped on board, saying that he would like to look at the machinery. After looking it over, he asked who had designed the engine, and, upon learning that the designer stood before him, bargained with him for a copy of the drawings. This copy Mr. Holloway forwarded in due time, and soon after received a telegram offering him a position in Pittsburgh, at a liberal salary, to superintend the construction of two steamships for the Atlantic coast-trade. Accepting this offer, he went to Pittsburgh, and, on entering the draughting-room of the works there, was surprised to find his drawings, endorsed with these words: "Approved. Horatio Allen, President Novelty Iron Works, New York." At that time the Novelty works were the largest and most noted machine-shops in the country, and Horatio Allen stood at the head of mechanical engineering in America.

It was about 1850 when Mr. Holloway designed and superintended the building of the two propellers in Pittsburgh, and

took them down the Ohio and Mississippi to New Orleans and thence around the coast to New York. In 1851 he designed and built, in Wilmington, Del., a side-wheel iron steamer for the Cuban trade. After this, he had charge for some time of the Cumberland Coal and Iron Company, at Cumberland, Md. From Cumberland he went to Shawneetown, Ill., where he remained in charge of large coal- and iron-works until about 1857, when he was called to become successively superintendent, manager, and president of the Cuyahoga Iron Works, which he conducted with conspicuous success until 1887, when the works were sold.

This was the conclusion of the most active part of his career; and indeed he longed for retirement and repose. But his willingness to work was greater than his desire to rest; and he remained in harness to the end, occupying for some years the position of consulting engineer with the H. R. Worthington Company, of New York City, and subsequently a similar position with the Snow Pump Company, of Buffalo, N. Y.

Although he had made his way as an engineer by the force of his own genius and hard work, and without the aid of a scientific school-training, Mr. Holloway, like many other successful self-made men, appreciated the advantages which he had not himself enjoyed. Yet he was loyal in his recognition of the training afforded by the school of experience. In an address before a graduating class of mechanical engineers at the Stevens Institute, he said of technical education:

"I have ever felt the want of it myself and have admired it in others; but for all that, I have a most kindly feeling for my 'Alma Mater,' the old-time country machine-shop, of which to-day there lies not one stone upon another. Not one sodden timber rests in the forebay, and only the notched recesses cut in the rocks on the river bank remain to show where the old water-wheel once slowly turned on its well-worn gudgeons; and much as I admire the grand engineering-schools of to-day, now that the hard toil of my youth is over, I look back to the old shop with a thankful heart, that amid its grime and dirt, amid its hard work and unpaid services, I learned of patience, of perseverance, of self-reliance, and handicraft; and above all, I learned under most trying and discouraging circumstances to close tightly my lips together and inwardly vow that I would make it go, somehow."

He took a hearty pleasure in social and professional intercourse, and was an active and honored member of various engineering societies.

Joining the Institute in 1875, he signalized his connection with it by memorable services as one of the hosts at the Cleveland meeting of that year. In 1887, and again in 1894, he was elected a Vice-President.

In 1880 he became a charter member of the Civil Engineers' Club of Cleveland, was elected Vice-President in 1882, afterwards three times President, and honorary member in 1891.

He was one of the founders, in 1880, of the American Society of Mechanical Engineers, and was for three years a Manager, and in 1884 and 1885 President of that Society.

The Engineers' Club of New York City also counts him among its founders. He was one of its Board of Management from the beginning, and served two years as its President, with honor to himself and to the Club.

This life, so full of the romance of endeavor and achievement, so rich in honor and affection, ended at last not far from the spot where it had begun. Mr. Holloway died in his own peaceful home on the banks of the Cuyahoga; but the river no longer flowed, as it had done seventy-one years before, through a frontier-wilderness. The growth of a great empire had transformed the scene upon which he closed his eyes, as he fell asleep among those who knew and loved him best. And in that mighty change none had wrought more potently than the pioneer-engineers of America, of whom he was an honored representative.

Mr. Holloway's generosity, tolerance and good-fellowship were known to everybody. The kindly way in which he delighted to speak of others was a revelation of his own character. John Fritz truly says:

> "It will be a most difficult task to do him justice. He was no ordinary man; and I know of no one who has left more true and sincere friends."

And it may well be added that the reason lies in the fact that he was himself a friend, true and sincere.

He was a bright and ready speaker, yet still more gifted with the pen. He wrote many articles and addresses, which were characterized by felicity, clearness and poetic feeling.

In illustration of these qualities, I give the following extracts from an article in *Locomotive Engineering*, entitled "Only a Picture:"

"It was a little thing—simply a small picture of an old-style locomotive placed in the middle of the page of a technical journal . . . . and a line below it which read as follows: 'Old Cuyahoga Engine, Built at Cleveland, Ohio.' . . . .

"The pictured engine might be the 'Reindeer,' 'Antelope,' 'Leopard,' or any other of the fleet engines designed by Ethan Rogers; but no matter which one it was, it was, in its day, a beauty and a runner. It looks light, the engineers of to-day will say. Well, it was light, and fortunately so; for the road on which it had to run was made of light iron rails, in many places spiked to slabs that lay on top of the ground, with neither ballast under them nor ditches beside them; and many a time did the engines come into the round-house, after heavy rains, clay-washed from truck to top of smoke-stack. Those new roads were not only unballasted, but they were so uneven that had not the engines been lightly built and of the best wrought-iron, they would have wrenched themselves to pieces on the roads they had to travel on.

"Those were pioneer days for railroads in Ohio. The few and newly-built roads were mostly through the woods and swamps, having a single track, with infrequent sidings, but with plenty of wet-wood stations in the winter and plenty of dry-water stations in the summer, and telegraph-lines at no time; but the engines—they were daisies! . . . .

"I suppose, if any of the engineers of to-day, the fellows who run the big moguls or the consolidations or the flyers on the Limited, should happen to see this picture, they would wonder among themselves what that curved arm near the air-chamber of the pump was for—that is, if they happened to know that pumps were at one time used on locomotives; and they would wonder why two valve-stems came out of the steam-chest. But you and I know that the curved arm worked the independent cut-off valve that Rogers put on the 'Cuyahoga engines,' and which helped to make them famous in their day; for the vim with which they would start a heavy train, and the economy with which they used steam while underway, used to astonish the down-east engineers who came out west, later on, with their heavier-built engines.

"Many and long were the disputes and discussions of the men who used to run and swear by the 'old Cuyahoga engines' as to their superior merits, as compared with engines brought from the Eastern shops and run on the same or adjoining roads; and oddly enough did they settle it. When differences now exist as to the superiority of one make of locomotive over another, the settlement of the question is left to scientific experts, who are usually professors of mechanical engineering in some college or technical school, who proceed to lash students to the front of the engine, one on each side of the cow-catcher, furnishing them with levers, pulleys, strings, indicators, stop-watches, etc., with instructions to take cards from the two steam-cylinders under the varying conditions of load, speed and grade. The observers come back from the trip with their hair full of dust and cinders, their faces marked with grime, and their hats full of slips of paper covered with curved lines, all differing from Hogarth's line of beauty. Over these curved lines grave professors then solemnly ponder, accounting as best they may for their sinuosity, and guessing at what they cannot explain; after which, with the aid of planimeters, scales and logarithms, they figure out that one engine is better than the other.

"Not so were settled the questions as to which locomotive could pull more, steam better, run faster or hang on longer, in the days in which flourished the old Cuyahoga engines. There were good talkers among the runners of those days, who were not afraid to express, in language often more expressive than polite, what they thought in favor of their own engines or in disparagement of others; and many a

summer day was made warmer as a group of engineers on the shady side of the round-house whittled, bragged and bantered each other. Once, after an unusually warm debate over the performance of a newly-arrived Eastern engine, as compared with a pet engine built at the 'old Cuyahoga,' it was decided to have a trial of the two engines in order to settle the matter.

"The consent of the master-mechanic having been obtained, a trial was arranged which, in every respect, differed from the trial-trips as now made and above described. What they wanted to know was, which of the two engines, having the same quantity of wood and water, could go the farther on the same day and over the same track. So it was arranged that the 'Cuyahoga engine' and the Eastern or Yankee engine, as it was called, should both start on an equal footing from Columbus, and run as far as they could towards Cleveland without replenishing. It may be well understood that each engine was put in the best possible trim, and each engineer and fireman was at his best. Along the line at every town were gathered the railroad-men, from the wood-sawyer to the station-agent, to greet and cheer their favorites as they rolled along northward, until, at last, the Eastern engine struck the descending grade several miles outside of Cleveland, and by its aid managed to crawl into the depot, bereft of wood, water and steam. Then the query was, Where was the 'Cuyahoga engine,' of which so much was expected; had it gone dead and cold somewhere back in the woods, and would another engine have to be sent out to drag it in, lifeless and disgraced?

"For a while it looked blue for the Cleveland boys, but not long; for soon their pet engine was seen bowling down the grade, and, as it neared the depot, the crowd parted to clear the track, when the engineer motioned to open the switch leading to the Lake Shore track. Then, with a defiant blast of victory, it dashed between the long lines of spectators, turned its front towards Buffalo, and, climbing the heavy eastward grade, the backwoods engine rolled on, and never stopped until it reached Painesville, thirty miles away, and, like Sheridan, won the day. Such a test would not at this time be deemed at all scientific, or perhaps satisfactory; but it settled the dispute years ago, when the trial-trip was run from Columbus to Cleveland.

"The shops from which these engines came were the first in which locomotives were built in the West; and they had few or none of the appliances with which the present locomotive-works are so well supplied. They were situated on the banks of the Cuyahoga river, with no tracks near, on which to place the engines after they were completed; and many a man would have shaken his head, had he been asked to build engines in such a shop and with such tools, and then to take them over a rickety pontoon-bridge in order to deliver them on a railroad track. But Ethan Rogers had the genius to manage it, and the pluck to dare it.

"What a time it used to be, when it was noised about town that Rogers was going to take a new locomotive over the bridge! and what a job it was to get it up out of the yard into the street, and to run it around there on an improvised turn-table! After this was accomplished, long timbers were laid across the old pontoon-bridge and a short distance on the opposite bank. In the meantime, steam had been raised in the boiler, and the crowd of spectators driven from off the bridge, and the street cleared for a run which might result in reaching the other side, or in sinking bridge and all to the bottom, just as luck, or skill, and the coolness of Rogers at the throttle might decide. At last the decisive moment is come; and, with a shriek that might indicate defiance or despair, the throttle is opened and the engine makes a dash at the bridge, which, feeling its weight, begins to sink deeper and deeper, as the spectators hold their breath and wonder why he don't go faster; but Rogers has

done it before, and he will do it again. Nearing the opposite end of the bridge, with the water behind him awash on the pontoons, and the sinking track showing a sharp up-grade before him, he pulls the throttle-valve to its widest notch, and the spurred engine, leaping as if for life, with a breathing exhaust that tells of the struggle it is making, climbs up from off the sinking bridge, and lands on the bank safe and triumphant."

We bid him an affectionate farewell as he departs to join those brethren of his and ours whom he used to recall with such glowing praise. He goes to a meeting of good fellows, who found one another out while they were in the flesh. And some day we, who remain, shall get our notice, and journey to a glad reunion—with Holley and Coxe and Holloway on the Reception Committee!

The Development of Colorado's Mining Industry.*

BY T. A. RICKARD, STATE GEOLOGIST, DENVER, COLO.

(Colorado Meeting, September, 1896.)

THE history of this State is that of one generation. Thirty-six years only have elapsed since the birth of that beneficent industry whose footsteps were the first to traverse the wilderness of the prairies and penetrate the solitude of the mountains. So soon is history made. The men who halted on the rolling plain where Denver now stands and gazed westward at the snow-clad ranges in eager questioning of their possibilities of golden wealth, have lived to see a noble city built where once their camp-fires burned, and have participated in the discovery of a magnificent series of productive mines amid the mountains which seemed at that time the Ultima Thule of their pilgrimage.

To those who investigate the workings of our complicated mining and metallurgical industries, the story of their evolution from humble beginnings will appear an instructive romance. The record that tells it presents features common to the growth of modern mining regions, but also bears aspects peculiar to

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* This paper was subsequently presented before the Colorado Scientific Society. The present version, however, is later in publication, and contains some additions.

those local conditions upon which the development of these industries is so essentially dependent.

In the summer of 1849 a party of seven Georgians were taking a herd of thoroughbred horses across the continent to California. They reached Camp Lyon, on the Arkansas, in October, and, meeting James Dempsey, a government guide, they were persuaded by him that it was too late to cross the mountains that season. His advice was followed, and, moving northward, they established a winter camp at the junction of Cherry creek and the Platte. Upon a sand-bar, lining the south side of the river, they built two cabins. During the closing months of that year they prospected the alluvial banks of Cherry creek, but did not penetrate into the mountain canons for fear of the Indians. Gold was found in several places, and particularly at a point 16 miles up the stream. From the feathers of the wild geese which they shot, they made quills, serving as receptacles for small quantities of gold-dust.

This party consisted of Dr. Russell and his brother, Green Russell, A. T. Lloyd, G. W. Kiker, Charles Kiker and P. H. Clark. Early in 1850 they crossed the range by the Bridger pass and went on to California.* They mined near Downieville and were successful. Mention was occasionally made of the gold found in western Kansas when on their way across the plains. The goose-quills were evidenced in proof of the story. In the spring of 1857 they, with others, sold out their interests in California and returned to Georgia. Before separating it was agreed among several of them that in the near future they would form a prospecting party to go to western Kansas and search for gold.

In May, 1858, eleven of them met at the Planters' House in St. Louis. In addition to the original seven there were present J. A. O'Farrell, three men of the name of Chastine and another called Fields. All save two were old Californians. Having organized, they went to Leavenworth by water and thence to

* These facts regarding the first discovery of gold in Colorado are not to be found in any of the so-called histories hitherto published. I obtained them in July, 1895, from Mr. John Andrew O'Farrell, at Boise City, Idaho. Mr. O'Farrell knew the Georgians above mentioned, while they were mining in California, and subsequently accompanied them when they made their second trip in 1858 to the then western part of the territory of Kansas. It is this second expedition that is usually considered to have led to the first gold discovery.

Camp Harney along the military road. Late in July they left this frontier post, accompanied by an escort of twenty men under the command of Captain Lyon. In August the party reached the log cabins on the Platte. The banks were at that time covered with the wild cherries which gave the tributary stream its name. As soon as camp had been established, they went to the places where Russell and his friends had found gold in 1849. Sufficient was found to encourage them. Prospecting parties were then organized. One of these went northward until they came to a mountain stream full of large boulders. They went up to the forks of Boulder creek. In a small basin on the left hand branch they found gold, and called the spot Gold Run. Another party went across the intervening ridges to Fall river, and over into Spring gulch. They did not descend into the valley of North Clear creek at that time, but crossed Quartz hill and found rich gravel at Russell gulch, named after the discoverer, Green Russell. It was too near winter to begin serious mining. They returned to camp. Six of the party went east to procure provisions, returning in the spring of the following year, 1859.

This was the year of golden discovery, By the close of 1858 rumors of rich diggings had crossed the plains, the rush had set in and crowds began to arrive. Among these came John Hamilton Gregory, who, with J. M. Cotton and his brother, William Cotton, went up Clear creek and discovered the outcrop of the Gregory lode on the 6th of May, 1859. This date is the birthday of Colorado's mining industry.

The discovery of the Gregory lode was immediately followed by that of other veins, whose production in the succeeding years made Gilpin county the leading gold mining district of the Rocky mountains.

In the meantime, bands of prospectors had scattered all over the neighboring hills, and were finding the gold depositories whose later developments made the counties of Clear Creek, Boulder, Gilpin, Summit, Park and Lake one great mining region.

Boulder was contemporaneous with Gilpin. In 1860 and 1861 the Columbia, Ni Wot, Horsfal and Hoosier veins were discovered and brought the Ward district into prominence. A dozen years later the tellurides of Gold hill were first recog-

nized in the ores of the Red Cloud mine. The recognition of their true character led to the successful exploitation of the rich ore-bodies of the Magnolia, Melvina, Slide and John Jay mines, and to the growth of the hamlets of Sunshine, Salina, Providence and Magnolia. The Caribou district was born when Samuel Conger found the outcrop of the Poorman lode in the last days of 1869.* The development of the Caribou vein began in the succeeding year.

The pioneers who followed up North Clear creek (or North Vasquez, as it was then called) and founded Gilpin's industry also wandered up the south fork of the stream and discovered the veins whose development gave wealth to Clear Creek county. In August, 1858, George Jackson did some prospecting about Vasquez forks, and in the winter of that year he penetrated to Idaho Springs and went up Chicago creek.† On the 7th of January, 1859, he found rich gravel. This led to active search and successful work amid all the other tributaries of Clear creek. The diggers followed the stream to its headwaters amid the snows of the main range, and discovered the veins above Silver Plume and Georgetown. The sluicing of the soft outcrops of certain veins served as a link leading from placer- to lode-mining. The Whale above Idaho Springs was one of the first so worked in 1861. In the upper country, near Empire, the gossan of the Griffith vein was successfully sluiced in 1859. The silver-ores were also recognized about that time, the Running lode, in Gilpin county, having attracted attention to the white metal because of the yield of a peculiar bullion, rendered intelligible only when the presence of silver had been determined. The silver-mining industry of Upper Clear creek grew to important dimensions in the decade succeeding 1870. The Pelican and Dives on Republican mountain were discovered in 1868, but did not commence active production until 1871. The Pay Rock was found in 1872. The mines of Sherman mountain—the Terrible, Dunderberg, Cory City, etc.—began to be energetically worked early in the seventies; in Lower Clear creek, John Dumont began operating the Hukill in 1871. In 1878 the Hukill and Freeland were purchased by J. B. Mackay and associates.

* For many of these data I am indebted to Fossett's *History of Colorado.*

† As related in vol. iv., of Hall's *History of Colorado.*

The mines at the head of Virginia cañon woke to life by the opening up of the Specie Payment in 1876.

In the summer of 1859 a party of gold-seekers followed the Platte from the foot-hills through its gateway into the South Park, and, camping on the future site of Fairplay, they crossed over to the western slope and descended the head-waters of the Blue river. Near the place now covered by the town of Breckenridge, Reuben Spalding sunk the first hole that disclosed the riches of the placers of the Blue. In the following year the gravels of its tributary, the Swan, were prospected. Alma and Fairplay, on the eastern slope, sprang into life as the result of the alluvial mining which then began a productive existence. It was not until 1880, however, after the exhaustion of the first-found shallow gold-bearing gravel, that the veins of Summit county were exploited. The placers of the Blue river and its tributaries have yielded about $35,000,000.

So were born the mines of Clear Creek, Boulder, Summit and Park counties. But more wonderful discoveries were in store. Leadville was yet to be uncovered.

Among the scattering bands of placer-diggers who spread over the ranges in 1859, one party followed the Arkansas and camped on Georgia bar. In the following spring they continued up the river and divided at the junction of California gulch and the valley of the Arkansas. In a little valley leading from Iowa gulch they stopped at noon. In breaking through the snow to get water for their coffee, the creek had been reached, and in the sand John O'Farrell found some gold. The pieces of porphyry amid the gravel reminded the discoverer of similar conditions observed on the Feather river in California. Little did he guess the significance of those porphyry fragments, or the enormous wealth which that rock covered on the neighboring hills. This was April 6, 1860. George Stevens and party came soon after. Their discovery claims were just above the site of the A. Y. and Minnie mines. Then sprang up a placer-mining industry which lasted for fifteen years, and was only obscured by the greater discoveries which ushered in an era of prolific silver mining.

In the early sixties good gold veins were found on Printer Boy hill overlooking California gulch. These mines, of which the Printer Boy, Five-Twenty and Pilot were the chief, were

productive for several years and foreshadowed the development —thirty years later—of the gold region rendered distinguished by the yield of the now celebrated Little Johnny property.

In 1874 W. H. Stevens and A. B. Wood came over the range from Fairplay, where they were mining, to build the Oro ditch. When examining California gulch, Wood found float consisting of carbonate of lead, and began digging on the south side, now known as Dome hill, on what was afterward the Rock claim.* He sank a little shaft through the drift, which covered the outcrop, at a point subsequently worked by an open cut. He found ore in place, but of low grade. This was in the fall of 1875. He made arrangements to have some work done that winter, and this led to the uncovering of the outcrop, across California gulch, up Iron hill. The next year the whole line of outcrop-claims was located on the supposed vein; and ore was taken in 1877 from the Rock claim to the smelter at Malta, which had been erected three years before to treat the ores of the Homestake mine.† Stevens got the Harrison Reduction Company of St. Louis to erect smelting works in 1877, and in the following year Mr. James B. Grant put up the establishment from which in later years grew the magnificent metallurgical industry of the Omaha & Grant Smelting and Refining Company. In 1879 Messrs. Anton Eilers and Gustav Billing erected the smeltèr which in later years became the property of the Arkansas Valley Smelting Company.

In 1878, also, George Fryer sunk a hole on a hill north of Stray Horse gulch, and found carbonate ore, uncovering in this act the great ore-measure which, as "the first contact" proved, was one of the most remarkable bodies of ore known in mining geology. A month later Rische and Hook happened to sink a hole where the contact approached the surface, and found the ore-body which subsequently became the Little Pittsburgh mine, and the foundation of Tabor's fortune. Other discoveries followed fast. That year—1878—Leadville's output exceeded in value $3,000,000. In 1887 it culminated in an output estimated to be worth $13,500,000. Up to date its yield has been $215,000,000.

* Facts obtained through the courtesy of Judge Weston, of Leadville, who was among the early pioneers of that region.

† Situated on the Saguache range opposite Leadville.

In the meantime other mining camps were being born. In the fall of 1872 veins were found near Rosita, in Custer county, and in 1877 the Bassick began to produce, creating the excitement of Silver cliff.

Among the progeny of the Leadville boom were Kokomo and Robinson, in the Ten-Mile district, whose checkered career belongs to later days.

In the meantime the mining industry of Colorado was going through the troubles of its early youth. The history of Gilpin county is so typical of this phase of its development, that I have reserved it for special description.

Following upon Gregory's discovery, other veins, subsequently famous, were found. On the 15th of May, the Bates lode was uncovered, on the 25th the Gunnell, Kansas and Borroughs. The Bobtail was discovered in June.* The early mining operations consisted of the removal of the soft decomposed croppings which were washed in the sluices after the fashion of ordinary placer-mining. The harder outcrops were crushed under trip-hammers until arrastras were introduced, to give way in their turn to primitive stamp-mills. By the 1st of July, 1860, there were 60 mills in operation. Everything proceeded serenely. But the gossan gave way to pyritic ore as the discovery-shafts penetrated deeper. The saving of the gold became more difficult. In spite of these drawbacks, the richness of the upper portions of the first-found lodes was such as to leave a handsome margin and maintain a steadily growing population. In the winter of 1863, and the spring of 1864, several mines were sold in New York and Boston. A stock-mania supervened, only to collapse suddenly in April. At this time also came the period of incoherent processes, with the promise of 100 per cent. extraction. The inexperienced chemist, with his revelation for cheap ore-reduction, continued what the stock-jobber had begun. The mining industry of Gilpin was crippled unto death.

At this time the easily amalgamated surface-ores had in many instances become exhausted, giving place to hard pyritic material, which refused to yield up its contained gold. Extraction in the stamp-mills continued to become worse. Many mines

* These details I owe to an admirable little book on the gold-mines of Gilpin County, by Samuel Cushman and J. P. Waterman, 1876.

were compelled to close down; others were operated at a ruinous loss of the gold in their ores. A depression fell upon the district, which was not removed until 1868, when a general revival began. The leasing of claims by working miners led to new discoveries, and the consolidation of adjoining territory diminished expenses. At this time the smelter came to the rescue of the baffled mill-man. In 1867 the Boston and Colorado Smelting Company was organized by Professor N. P. Hill. In June the first experimental plant was erected at Black Hawk. In January, 1868, the establishment opened for business. In 1873 the company ceased the shipment of mattes to Swansea, and erected a refinery under the direction of Mr. Richard Pearce. In 1878 the smelter moved to Denver.

This represents a stage of progress common to all our mining regions. Crude milling-methods give way to fire-reduction processes, and the latter, by their heavy charges, invite the mill-man to improve on his cheaper methods, so that competition is restored. The Black Hawk smelter saved the mining industry of Gilpin county when it was on the verge of utter collapse. And on the restoration of prosperity, the owners of the stamp-mills were enabled to carry on experiments whose expense was met by the sale of ore to the smelter, so enabling them to evolve a method of stamp-milling which was well adapted to the treatment of the heavy pyritic ores produced by their mines. In 1871 the problem had been solved, and to-day 500 stamps do excellent work with the low-grade ore of the district.

Gilpin county has produced about $68,000,000 to date.

While these problems were undergoing solution, mining was winning fresh territory southward, amid that great complex of mountains whose waters drain into the San Juan river. A party of pioneers, guided by Jim Baker, crossed the Sangre de Cristo range and reached the head-waters of the Animas in 1861. In spite of snow-slides and Indians the search for gold and silver was extended over the neighboring ranges. The ratification of the Brunot treaty, in 1873, marked the cession of this part of the territory by the Indians, and removed one of the most serious obstacles to the development of the region. In the meanwhile, mines were being opened up on every side. The Baker party tested the river gravels, and the evidences of their placer-workings still remain in many a secluded valley to

tell of these first beginnings. In the spring of 1871 lode-mining may be said to have commenced by the discovery and location of the Little Giant vein, just above the present town of Silverton, by Miles T. Johnson. In the following year an arrastra was built, and the gold thus extracted out of the ore was taken for sale to Conejos, the nearest trading station. In 1874 Judge Green, of Cedar Rapids, Ia., commenced the erection of a smelter, the machinery of which came on burro-back from Colorado Springs, which the Denver and Rio Grande railroad had just reached. The ore supply of this smelter came principally from the Aspen, which at that time was the chief mine in the locality.

In 1875 Mr. J. A. Porter, the metallurgist of Green & Co.'s smelter, introduced the siphon-tap,* and in the following year he erected the first water-jacketed furnace† built in Colorado.

The development of this region soon rendered the Green smelter inadequate; and in 1880 the works of the San Juan Smelting and Refining Company were erected at Durango, which locality is the center of fields of excellent coking-coal. This was one of many instances, occurring at this period, of the tendency to centralize the smelting industries of the State in the valleys, where the confluence of the railroads from the adjoining mountains enabled the metallurgist to obtain the necessary mixture of ores. Thus, in the course of time, the large reduction-works of the State became concentrated at Denver and Pueblo, with Durango and Leadville as subordinate centers.

The prospectors who made Silverton their headquarters scattered up the valley of Mineral creek, and on the watershed separating this tributary of the Animas from the Uncompahgre found the veins which gave fame to Red mountain. In 1879, Charles Newman and Harry Irving located the Carbon Lake claim, and did their annual assessment until 1882, when a discovery of copper-ore was made on the adjoining Congress claim and shipments began. In August of the same year, Andrew Meldrum, while out hunting, stumbled upon the outcrop of galena which marked the now famous Yankee Girl

* The first siphon-tap in Colorado was put in by Arents, at the Swansea works near Denver. Mr. Porter had seen it in use at Eureka, Nevada.

† This water-jacket was round and 3 feet in diameter. It was made by Fraser and Chalmers.

vein. The American Belle and Guston were discovered shortly afterwards. Five years later this district became a magnificent producer of very rich copper-ores. In 1894 a smelter was erected at Silverton to treat the product of the region.

In the meanwhile, Gus Begole had crossed the western range from Silverton, and descending into the valley of the Dolores, located claims which later became the Yellow Jacket and Aztec mines. But the ore was too low in grade and the work soon ceased. In 1878 John Glasgow and Sandy Campbell came northward from La Plata City, and began the successful development of the Grand View and Atlantic Cable, causing the growth of the town of Rico. The news was spread abroad that another Leadville had been found, and crowds trooped in across the hills during the summer of 1879. In the fall of 1880 the smelter began operations. Nevertheless the district would have gained but slight distinction, had not the ore-deposits of Newman hill been found. These began to be productive when, on the 6th of October, 1887, David Swickheimer struck the big ore-body of the Enterprise mine.

In June, 1870, gold was found in Wightman's gulch. This led to the location of the Little Annie in September, 1873, and the opening up of the Summitville district, which is tributary to Del Norte, whose position made it a natural gateway to the watershed of the Rio Grande.

In 1874 the placers of the San Miguel river attracted the prospector to the slopes of the Ophir and Mt. Sneffels ranges. This led to the discovery of the Smuggler-Union vein, which is credited to John Fallon, who located the Sheridan claim in 1875. The next year J. B. Ingram located the Smuggler.

A French company, under the direction of M. Charles Laforgue, erected the present Pandora mill and operated the Pandora and Oriental mines during 1877 and succeeding years.

Among the early locations were the Belmont and Tomboy, situated on the trail which crosses the range from Silverton and Red Mountain, but the value of the lode was not evidenced in the first workings, and the company organized in 1892 by Mr. J. H. Ernest Waters was unsuccessful. His good opinion of the value of the mine, however, was amply confirmed by later developments, following a reorganization effected in 1894. The Tomboy mine became a larger producer two years later.

The fact that most of the mines were well above timber-line (11,000 feet), with the attendant high cost of transport and living, delayed the opening up of this, the Telluride district. The consolidation of claims and the energy of a group of very capable men overcame these obstacles, so that to-day the district offers some of the very best examples of the application of modern skill to mine-exploitation.

Beyond the southern ridges is the Ophir district, which became productive at about the same date. The Summit, one of the best producers in the early days, was located in 1874. It was extensively worked in the early '80's, and led to the erection of a smelter at Ames, in the valley of the San Miguel. The fuel used was aspen and spruce pine. Iron-sinter, from the springs above the town of Ophir, served as flux. The lead-ores came from the Summit, and also from the Caribou, Valley View, Nevada, Silver Belle and other mines. The campaign was brief; and all that now remains is a small slag-dump.

The similar fate of many such schemes of twenty years ago led to the gradual recognition of the futility of erecting small smelters in remote corners, where the needed supply of suitable ores and fluxes was unobtainable, fuel was expensive, and business conditions were unfavorable.

Among the old districts recently revived is the La Plata mountain region, north of Durango and south of Rico. In 1873, Captain John Moss, representing Tiburcio Parrott and other San Francisco capitalists, came up the San Juan river from Arizona and penetrated the La Plata mountains, being attracted thither by the gold-bearing gravel of the streams. He followed the latter to their source and discovered a large number of veins. The Comstock, Morovitz, Euclid and Ashland claims were located at that time. Parrott City was founded, and great activity characterized the camp for a brief period. But the complex telluride ores proved refractory; and the arrastra was found to be powerless to extract the values. Great expectations had a sequel of small accomplishment. The district became depopulated. In 1894 new discoveries were made, and a revival took place, leading to more serious work, which now promises better things.

And so we come to recent times, no less stirring than the old. The history of the last decade centers round the discovery of

Aspen, the stories of Creede and of Cripple Creek, the decline of silver-mining and the development of new gold-fields.

The first discovery in the Roaring Fork district, of which Aspen is the center, was made July 3, 1879, when Philip W. Pratt and Smith Steel, coming from Gothic by the Maroon pass, found the Galena mine on West Aspen mountain. On the following day they located the Spar claim on Aspen mountain, and on the 5th Messrs. Allbright and Fuller located, at the foot of Smuggler mountain, the Little Rock claim, covering a part of the property of the present Smuggler mine. The Smuggler claim itself was located August 30th, by Charles Bennett. On account of the theft of the first four pages of the district recorder's book, it became necessary to make relocations, which now appear on the records at Gunnison, the district being then a part of Gunnison county.

The first mineral survey was made on the Monarch mine, October 12, 1879, by John Christian, of Leadville. The site of the present city of Aspen was devastated, in September of that year, by a forest-fire, in which many horses and pack-mules were lost. An Indian scare, following the Meeker massacre, caused most of the prospectors to leave the camp; but a sufficient number remained to build log-cabins on the site now occupied by Aspen, which was located as a placer-claim September 20, 1879, by Walter Clark.

It was not until ten years later that the new district won a commanding position. At that time (1889) the Aspen, Aspen Compromise and Compromise mines maintained a large output, and it was in 1891 that the big bonanza of the Mollie Gibson was uncovered. Aspen is credited with a production of 8,275,000 ounces of silver in the year 1892.

N. C. Creede found the float of the Holy Moses vein, on West Willow creek, a tributary of the Rio Grande, in 1889. As a consequence, the King Solomon district, as it was then called, began to attract the prospectors scattered in the mountains above Del Norte. No important results ensued until in June, 1891, Mr. D. H. Moffat and Capt. L. E. Campbell came to Wagon Wheel gap to visit the Holy Moses, on which they had secured an option. Creede was engaged to prospect for them. Shortly afterwards, Theodore Renniger found rich float on Bachelor mountain. He was unsuccessful in finding the vein in place until

Creede came along, and, at a point 200 feet higher up on the hill-slope, discovered the outcrop of a large lode. He located the Amethyst claim and then informed Renniger, who on the same vein took up another claim, which he called the Last Chance. This was on the 8th of August, 1891. It should be added that, three years previously, J. C. McKenzie had located several claims on a heavy quartz outcrop, about 150 feet above the Amethyst. These were afterwards abandoned until the fall of 1891, when the Del Monte location covered them. *Débris* from the upper portions of the mountain had so obscured the outcrop of the Amethyst that its earlier recognition had been prevented. After Creede's discovery but little work was required to prove the existence of a magnificent lode, and, as a consequence, the camp sprang into tremendous activity, culminating in the boom of 1892. The immediate extension of the railway from Wagon Wheel gap stimulated a production which reached its maximum in an output of $3,100,000. Then in the summer of 1893 came the sudden fall in silver and a collapse from which Creede has not yet recovered.

Creede and Cripple Creek were rivals in attracting attention in 1891. Both have had strange vicissitudes. Our new gold-field lies on the southern slope of Pike's Peak, whose snow-clad crest was the beckoning guide of the pioneers of 1858. Although no gold-discoveries of any moment were made in the early days among the streams, or on the hills lying at the foot of the old beacon mountain, it nevertheless gave its name to the mining excitement of that period. Time has, however, of late, justified the expectations of the tenderfeet of forty years ago.

The first recorded locations were made in February, 1891, but the clustering hills which are now dotted with productive mines, had been disturbed by the miner's pick as early as 1874. Silver-ores were found in a shaft located close to the present Elkton mine. Ores rich in the white metal have been found in late years, and the Moose made one shipment of 30 tons, carrying an average of 70 ounces of silver, in addition to the gold.

It was in April, 1894, that this district became the scene of the queer fiasco which has gone into local history as the Mt. Pisgah excitement. A crowd of 4000 men were brought together on the rumored discovery of rich placer-ground. Nothing was found save in the holes of the original

locators. Man had endeavored to remedy nature's niggardliness, and the poor rock had been artificially enriched. Lynching was threatened, but the attempt to catch the perpetrators resulted in failure, after which the hill slopes again became the quiet cattle-ranges for which they seemed best adapted. Those deluded prospectors let slip a great opportunity. Mt. Pisgah's dark front now overshadows the busy streets of the town of Cripple Creek; and, on the ridges opposite, line after line of smoking chimneys bespeak a long succession of productive mines.

During the spring of 1891 Cripple Creek began to receive respectful mention in mining circles; but the discoveries made at Creede during that summer diverted attention from a district whose previous experience had given it an unsavory reputation. When, however, the silver-market collapsed in June, 1893, and mining seemed prostrated, the men of Aspen and Leadville turned with the energy of despair to the new gold-field which previously had been pooh-poohed, and the concentrated activities of the State were directed to the development of Cripple Creek. With a rare good fortune, the new camp answered to the call, and, as explorations extended, there came a swift succession of rich discoveries which caused the output to spring from $583,000 in 1892 to $2,100,000 in 1893. This growth continued so that in 1894 the yield was $3,900,000, and in 1895 it reached $7,800,000. These figures are sufficiently eloquent of the development of a district which to-day is the largest producer of all the gold-mining camps of the United States.

During the past three years discoveries in other parts of the State have received mention, and large claims have been made in behalf of several new districts. West Creek, Cottonwood, Hahn's Peak and the Gunnison region may be cited. Of these the last named is much the most important. At the head-waters of certain streams, tributary to the Gunnison river, there have been found veins which are now supplying several stamp mills with pay-ore, and have given birth to the settlements of Vulcan, Spencer, Iris and Dubois.

While the new territory which has been won during later days adds its tribute of gold and silver to the yield of the old established mining centers, it must be noted that the latter have

also accommodated themselves to new economic conditions. In certain instances the gold-output now exceeds the silver, while before 1893 the reverse was the case. This is typical of the changed conditions regulating the industry. It is a striking evidence of that resourcefulness which has enabled the State to overcome difficulties. The decline of silver-mining led directly to an impetus in the search for gold, and the variety of ores so puzzling in the infancy of our smelting industry is to-day its chief aid, because it permits the attainment of that admixture of material which is the essential of successful reduction.

Colorado has yielded to date gold valued at $137,475,000, and silver having a coinage-value of nearly $400,000,000.

Thus, from humble beginnings, a great and complicated industry has been created. Its development may be summarized in four periods: the discoveries in Gilpin county and the adjoining camps in the granite rocks of the Front range; the era of silver-mining in the carboniferous limestones of Leadville, Aspen and Rico; the development of the fissure-veins in the andesites of the San Juan; and lastly, the revival of gold-mining consequent upon the uncovering of a great series of ore-deposits in the volcanic complex of the Pike's Peak region.

The information contained in this paper was largely derived from those who took a prominent part in the early development of the State. The writer is particularly indebted to Judge Weston, of Leadville, and to Messrs. T. A. Porter, J. P. Waterman, J. A. O'Farrell, Charles Newman, Major Campbell, H. A. Vezin and D. W. Brunton.

The Occurrence and Treatment of Certain Gold-Ores of Park County, Colorado.

BY B. SADTLER, DENVER, COLO.

(Colorado Meeting, September, 1896.)

THE oldest producing district of Park county, and in fact one of the oldest producing gold-districts of the State, is situated on the head-waters of the Platte and tributary streams. Geo-

logically, the district is a continuation of the Leadville series; the greater portion of it, in fact, being mapped in the Leadville sheets accompanying the work of Prof. S. F. Emmons, of the United States Geological Survey. It differs, however, from the Leadville district proper, from which it is separated by the Mosquito range, in having a uniform dip toward the east, in being far less faulted, in showing a very much smaller amount of porphyry and in being very much less decomposed, the tremendous decomposition observed in the Leadville limes and porphyries being only comparatively slightly shown here. The London fault seems to be practically the only large fault showing east of the crest of the Mosquito range; others observed being comparatively small. The occurrences of ore mentioned below present, however, some very distinctive features, which are briefly as follows:

The principal gulches, of creeks heading in the Mosquito range, have cut down into the Laurentian granites and schists. The mines in this section, which include the Horseshoe, Mosquito, Buckskin and other gulches, are almost entirely worked by tunnels and above the creek-levels, the Brownlow and one or two other properties being the principal exceptions.

The larger proportion of the veins observed have not been developed at all in the granite, and some of them do not extend down into the granite. This is probably due to the fact that most of the veins mined in the quartzite have become pinched on entering the granite, and some of them seem to start right at the contact between the quartzite and granite, probably receiving their mineralization from this contact, or from different planes of parting in the sedimentary rock. There are, however, some notable exceptions to this statement. Among those coming under my personal observation, the Cleaner, Chicago and Union No. 5 show ore in the granite of grade equal or superior to that encountered in the upper levels.

A generalization of the ore-occurrence would be roughly as follows: Where the veins occur in the granite they show all the ordinary characteristics of veins in this formation; on reaching the quartzite they generally widen out and do not seem to have well-defined walls, protrusions of the quartzite into the ore-body being frequent, and the ore in many instances following the bedding-planes for some feet away from the vein. This

is very clearly observed in the workings of the Fassett Mining and Milling Company, and, in fact, in practically all of the veins in the quartzite. It is especially notable in the Criterion mine on Mt. Bross. The greater part of the ore exposed in this property lies along the contact; a sheet of ore, 6 feet and more in thickness, being shown by both the outcrop and drifts for at least 75 feet away from the vein.

Immediately overlying the Cambrian quartzite the ore-bodies are generally considerably extended in a lateral direction along a band of shale which underlies the Silurian white lime. At this point the veins which have been heretofore mainly gold-bearing, and, in the mines at any distance above creek-level, pretty well oxidized, commence to show a considerable proportion of galena and silver, in addition to the gold. The veins also change their appearance, being comparatively narrow and showing well-defined walls, except at the intersections with some of the planes of parting, where they again spread out, forming in cases quite considerable ore-shoots at these intersections. Continuous development or underground connections between these veins and the ore-bodies found in the Carboniferous lime, have not been made; but the observations made in the Moose, Dolly Varden and other mines in the Carboniferous lime seem to indicate that, as a rule, these fissures do not penetrate an intrusive sheet of porphyry which is generally found near the horizon of the parting quartzite. Such ore-shoots generally carry high silver-values, and seem to resemble, in their appearance, the Leadville ore-bodies, overlying each other on successive contacts without any well-defined connecting fracture or fissure.

The general trend of the fissures observed has been northeasterly and southwesterly. They are at times intersected by porphyry dikes, which have in several instances apparently acted as dams, showing on the southerly side an enlargement of the ore-body, while toward the north the veins are usually pinched for some distance beyond the dikes, rarely commencing to widen again inside of fifty feet.

The ore in the quartzite and granite, especially in the former, has been generally found to be oxidized; this oxidation being more complete in the mines above the creek- or gulch-level, and a larger proportion of sulphides showing in each lower

level reached. A vein, however, which is thoroughly oxidized in the quartzite, will frequently show a very large proportion of galena and other sulphides in the limes. I would attribute this mainly to the greater width of the vein in the quartzite and to the greater opportunities for percolation, due to the excessive shattering of this stratum.

As we proceed further east, we reach the rocks of the Jura-Trias, which show in Mt. Silverheels. Several properties on the eastern slope of this mountain were investigated, and, as far as observed, the ore was found mainly to occur along the bedding-planes of the sedimentary rocks, although some was also noted along the contacts of porphyry dikes intersecting these rocks. The observations made in regard to these properties seem to apply, as far as I can learn, to other districts along the eastern slope of the Mosquito range, which I have not visited.

Southern Park County.

The area of recent eruptive rocks observed at Cripple Creek is mapped by Hayden as extending across the southern end of Park county, one corner of which in fact reaches within a few miles of Cripple Creek. Considerable prospecting is going on in this section, some of which I have had opportunity to observe; the results so far obtained, while encouraging, have not as yet resulted in the discovery of any producing mines. They are briefly as follows:

At and near Balfour a number of well-mineralized porphyry dikes were discovered and some float of good grade was found. The miners made the mistake of considering these dikes to be veins, and sunk on them in a few cases, finding, as was to be expected, only occasional seams of good-grade quartz. The contacts of these dikes were in no cases prospected—which was doubtless a mistake, as the dikes themselves generally showed from $2 to $4 in gold per ton. About half-way between Balfour and Cripple Creek, near the eastern edge of Park county, or, to be more accurate, in the southeastern corner, considerable systematic prospecting is being done. Part of this has been done under my observation, with the following results:

A series of dikes of igneous rock were observed to carry from $2 to $6 gold per ton. Prospecting the contact of these dikes, mainly, as yet, by 10-foot holes, has shown notable amounts

of low-grade ore, running from $6 to $12 per ton. Besides this, several bunches or pockets, yielding between $20 and $30 per ton, have been encountered, as well as two small pockets of very high grade; assays of from $100 to over $1400 per ton, all in gold, having been obtained. I have also information from reliable parties in Cripple Creek, who are having work done in the same section, with similar results so far—*i.e.*, considerable low-grade, and occasional pockets of high-grade, ore. There has not as yet been sufficient development in this district to speak with certainty, further than the facts quoted above.

Through the kindness of Prof. R. C. Hills, of Denver, who determined it for me, I am able to describe the dike-rock above mentioned, which is a mica-porphyrite, with the following characteristics:

Macroscopic.—Color purplish-gray, showing phenocrysts of biotite and plagioclase.

Microscopic.—Moderately fresh plagioclase in prominent crystals, with biotite crystals somewhat smaller, in a granular ground-mass, in which recognizable microliths of plagioclase and biotite are imbedded. The ground-mass is rather murky, owing to a little kaolinization induced by weathering.

Ore-Treatment.

The greater part of the ores examined near Alma required rather more than simple amalgamation. The ores found in the quartzites were most nearly free-milling, when found in the upper part of the mountains. Beside what was saved by amalgamation, these produced about 3.5 per cent. of a concentrate, carrying about $30 in gold per ton. Ores taken from mines lower down, or nearer the creek-level, in the quartzite, showed an increased proportion of sulphides, yielding from 10 to 18 per cent. of concentrate. As we approach still nearer the creek-level the proportion of sulphides runs still higher. While chlorination is not the rule in the district, concentrates from the particular mines tested were all susceptible of chlorination, tests having shown from 80 to 90 per cent. saved by that process. While the ore from a number of properties not examined is reported to be, and probably is, of the same character, a considerable proportion of it certainly carries copper; some of it having been made a basis for copper-smelting operations

in the earlier days of the camp. The ores occurring in the limes, where they are not high enough in grade to stand direct shipment to the smelters, concentrate very nicely on account of their percentage of galena. This concentration is especially necessary where the ores are mined together with those from the quartzite, jigs being used to remove the galena prior to the amalgamation which is in most instances necessary, on account of the general presence of free gold in the quartz, independent of the sulphides or oxides.

In most of these cases some rusty gold seems to escape in ordinary mill-work. Experiments for the recovery of this, by scouring and grinding such tailings, have been extremely successful when a sufficiently fine mesh was used; and devices to effect such a saving will be put in on a large scale.

The ores from the Silverheels district referred to seem to be susceptible of amalgamation only near the surface, and to depend on concentration, or sufficient values for shipment, for profits in deeper mining. In the southern part of the county laboratory-tests and small mill-runs have been made on the low-grade ores referred to, both by chlorination and cyanide processes, with favorable results.

The Occurrence of Gold-Ores in the Rainy River District, Ontario, Canada.

BY WILLIAM HAMILTON MERRITT, TORONTO, CANADA.

(Colorado Meeting, September, 1896.)

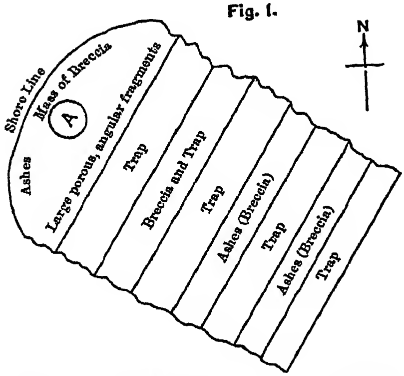
SINCE 1880 the occurrence of gold-ore has been known in the Lake of the Woods district. In 1879 the writer examined a so-called silver-location in the lake. More recent developments have extended, south and east, to the Rainy lake and along the Seine river, which flows from the northeast into the Rainy lake.

The gold occurs in quartz-veins in the Huronian series. These Huronian rocks occur in belts in the Laurentian; the latter being the predominant formation in the eastern part of Ontario.

The general characteristics of the Huronian rocks in the districts above mentioned are the evidences they carry of great igneous activity. On some of the islands in the Lake of the Woods volcanic ashes have been weathered out most clearly, and lava beds, with which they are interstratified, are equally well-marked. Figs. 1 and 2 show these conditions. Most of the igneous rocks in this locality have been subject to considerable alteration.

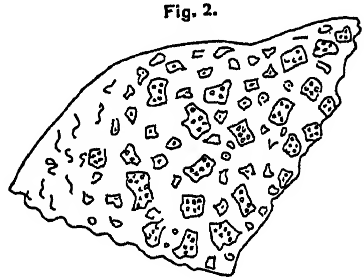
In Schists.—The majority of the volcanic rocks, both eruptive and intrusive, have been highly altered by pressure and chemical action into a series of schists and schistose conglomerates, chiefly of a greenish color. These schists are occasionally acccompanied by quartz-veins running with them. The quartz

Fig. 1.



Outcrop of Volcanic Series on Crow Island, Lake of the Woods, Ontario.

Fig. 2.



Sketch of Volcanic Tuff Weathered Out at A, Fig. 1.

is, for the most part, very irregular in these occurrences, and is generally lens-like in its disposition in the schists. There are exceptions, however, to this rule; as, for example, at Bath Island, where remarkably persistent veins run with the apparent bedding of the schists. (See the description of Rock-section No. 1, under *Examples of Country-Rocks*, page 861.)

It is noticeable in some instances that the bedding of the schists may, indeed, have been set up by pressure after the formation of the veins, or after the action of the forces which gave rise to the fault- or fissure-lines in which they have formed. In two uncertain instances from the Lake of the Woods district, and in another very conclusive one from British Columbia,

I have discerned in rock-sections a line of bedding previous to that now adopted by the schistose structure running parallel to the vein.

The minerals in the Lake of the Woods schists have been much crushed, and a great deal of alteration has taken place. Flow-structure and fracturing of the mineral is very commonly noticed in specimens of the wall-rocks. In many instances the quartz infiltrations and bands in schists have, without doubt, followed shear-zones, and the quartz-infiltrations occupy the buckles of the formation, as is frequently noticed in the Cariboo schists of British Columbia. It is needless to say that quartz-bodies occurring in this way are very irregular and terminate abruptly; but they are so numerously distributed in some bands of the formation that mining, without doubt, will be prosecuted profitably where the quartz-lenses occur at no great interval from one another.

In the Seine River district, stringers of quartz, carrying excellent specimens of free gold, are found running with the schists in many places, and it is confidently expected by the owners that the belts in which they occur will be mined with profit. This occurrence is probably similar to some of the low-grade rock which is mined in North Carolina.

There is little doubt that the schists, especially in the vicinity of the intrusive granite, are well worthy of attention; and it is altogether probable that belts of schists and quartz-stringers will be milled profitably on a large scale in one or more localities.

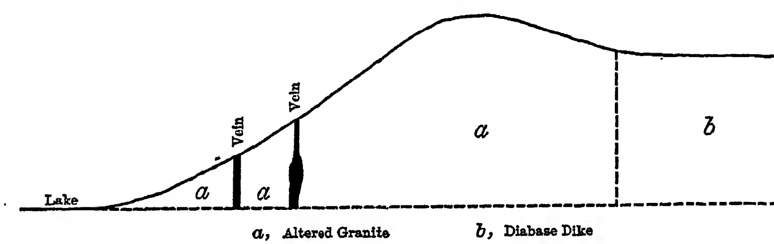
No locations in the schists have had much work done upon them in the Seine River district. The gold which has been found on the surface occurs essentially in a free-milling condition. Rock-sections Nos. 2 and 3, under *Examples of Country-Rocks*, below, are other representatives of these schists in which veins occur.

In Granites and Greenstones.—The most prolific of all rocks in the Rainy River district is intrusive granite, both in its normal condition and altered to the decomposed state of protogene. The granite areas in the Lake of the Woods part of the district are chiefly of the former character. Sometimes the gold-carrying veins occur at or near the junction between granite and greenstone, and sometimes they are found cutting through the

granite into the neighboring greenstone. It is noted that the copper contents have increased when in the latter rock. Frequently the schists in the vicinity of the granite carry veins which furnish a high-grade of auriferous quartz.

There are also areas of plutonic greenstones which retain more or less of their massive characteristics, like the intrusive granite. These are also found to contain quartz-veins, which carry more or less gold. This is true both of the close-grained or greenstone type, as in the case of the country-rock at the Pine Portage mine (Rock-section No. 4, below), and of the coarsely crystallized gabbro type, which is developed in a large area in the vicinity of Lake Bad Vermilion, where the Randolph and other veins carry some high-grade gold-quartz.

Fig. 3.



Section of Formation at the Sultana Mine, Lake of the Woods, Ontario.

One very interesting occurrence in the gabbro near Lake Bad Vermilion is worthy of note. A large vein of quartz carrying breccia of the wall-rock has again been subject to fracture, chiefly at right angles to its strike, and a great number of veinlets cross it at right angles and carry little breccia and horses of the original vein. These latter veinlets furnish fine specimens of quartz-gold, and individually give high results. An attempt is being made to ascertain if the whole mass will furnish sufficiently high results to make it a paying proposition.

The same gabbro on the other side of the lake, near a large granite area, carries strong quartz-veins which sometimes yield as much as $25 to the ton. These veins have a good deal more copper in them than those in the neighboring granite. Fig. 4 shows the series of formations in a generalized section.

Plutonic dikes are found intimately connected with the origin,

if not with the mineral contents, of some of the veins. This is notably the case at the Sultana mine, in the Lake of the Woods district, as shown by the diagram, Fig. 3, where it will be observed that a great dike has fractured a porphyritic quartz-syenite, or a granite, which has been squeezed and very considerably altered, and in which occurs the Sultana vein, running parallel with the trend of the trap (Rock-section No. 5, below). The vein being mined varies from a foot or two up to 40 feet in width, and its mineral contents are chiefly iron pyrites with free gold, milling as high as from $20 to $30 per ton.

It has been stated in some official reports and bulletins that some of the gold-veins occur in the original Laurentian gneiss or granite. I think that this will be found, on thorough investigation, to be a mistake; that the granites in question will be

Fig. 4.

a, Conglomerate (Keewatin)
b, Intrusive Granite
c, Keewatin (Huronian) Green Schists.
d, Intrusive Gabbro.
e, Laurentian Gneiss.
f, Auriferous Veins.

Cross-Section of Formations, Illustrating the Occurrence of Some Gold-Bearing Veins near Seine River, Rainy River District, Ontario.

found to be intrusive, and that the semblance of gneissic or schistose structure, observable near the vein, has resulted from local pressure and metamorphic action. The Sultana vein is an example.

In the Seine river section, the most important veins yet developed are in fissures cutting areas of greenish altered granite or protogene (Rock-section No. 6, below). Mining operations are in progress on several of these. In some of the areas the veins are strong and very numerous, but only a small proportion of them carry enough gold to be mined with profit.

A recent discovery in the altered granite, improperly called a dike, appears to be a large quartz lode, much mixed with altered green chloritic material, and with some finely diffused iron and copper pyrites. On it, the Hammond-Folger locations have been proved to carry some good pay-ore; and it is said that the workable pay-quartz is of great width.

In two well-defined instances, at the Ferguson and William Wiegand locations, crushed or altered country-rock, alluded to as a dike, runs with the vein. Schistose structure has been set up in it. Its composition consists of large pieces of quartz, fine grains of feldspar and muscovite-talc ground-mass, with some biotite and accessories of pyrite and calcite. In appearance it is a gray close-grained rock, differing distinctly from the usual protogene.

In Felsitic Bands.—In the Lake of the Woods and Seine districts bands of schistose felsite, composed chiefly of feldspar with a little mica, occur. They are generally sprinkled with small cubes of iron pyrites (Rock-section No. 7, below). These bands have received a great deal of attention at the hands of prospectors, for frequently they are found to pan a small amount of gold, as a rule just enough to prove an aggravation. Veins of quartz are also found cutting them, and these sometimes carry enough gold to be found profitable; but, again, some of the veins cutting this formation, while mineralized with a good deal of iron pyrites, are yet not rich enough to work profitably.

General Character of Ore.—The gold-bearing quartz in this old formation might naturally be expected to be peculiarly refractory. Such, indeed, was the impression resulting from the first operations which took place in the Lake of the Woods, and from the first investigation by the Geological Survey of the character of the ores. Much arsenic was reported to have been found, and the gold-values were said to be particularly associated with refractory pyrites, from which they could not be extracted by amalgamation. A couple of mills were started some years ago in the Lake of the Woods district, and the result of their work was very detrimental to the district. Either they ran on an ore which was of too low grade to pay, or ignorance of the principles of milling was the cause of their inability to save the gold.

Recent operations have proved that, in one instance in the Lake of the Woods, and in another instance in the Seine River district (both in altered granite), the ore is free-milling at the depth of more than 200 feet; and, in a third instance, that while the ore is free-milling, yet more skill than is at present brought to bear is needed to save fine gold which escapes in

the tailings. Cyanide-treatment is about to be used on these tailings, and it should be successful.

To quote a few examples: The result of the working of the Sultana mine up to the present time has been that at least 90 per cent. of the ore has been free-milling. The concentrates, of which from 1 to 2 per cent. are reported to exist in the ore, average from $30 to $40 per ton, and are treated by chlorination. A mill-run on the Seine river, from a granite (protogene) area, gave a similar proportion of yield.

As an example of a high yield from a small lot of ore, a mill-test of 114 tons from the Micado mine in the Lake of the Woods district might be quoted. This yielded by free-milling $7640, or $67 to the ton, the concentrates not being estimated.

Pan-amalgamation tests from a vein in the protogene gave a little over 90 per cent. of free-milling ore, and the percentage of free-milling extraction from ore occurring in the schists, from near the surface, was also very high.

It must be noted that, with the exception of the Sultana and the Regina in the Lake of the Woods district and the Foley mine in the Seine River district, no considerable depth has yet been reached; but as the above have proved their ore to be entirely free-milling from 200 to 300 feet in depth, the outlook for the district is extremely promising as an area in which free-milling ores can be mined at a comparatively low cost. It might be expected with reason that the free-milling character in the altered granites would last as far down as the granite has been altered into protogene.

In the Seine River district the veins cutting protogene are found to carry their gold in a very free condition, and in most cases the gold is very coarse. Some ores which carry much pyrites and which might be expected to be more refractory, have shown by milling-tests that they have, as far as sunk upon, the major part of their gold in a free condition.

Some of the veins cutting the protogene are highly mineralized with zinc-blende, iron pyrites and galena, and to a lesser extent with copper pyrites.

The gold-contents do not appear to be influenced by an excessive mineralization; for some of the basest-looking ores, highly charged with the above-mentioned minerals, have proved on assay to run comparatively low in gold-contents,

while other veins bearing the same amount of mineral give high results.

It is noted, however, that in cases where mineralization is entirely absent a vein is usually found to be barren in its gold-contents, following the general rule in this regard; and it is also observed that the ore-shoots which carry gold enough to pay to mill are by no means universal, though their proportion is probably equal to that in most other mining districts.

There is little doubt in the mind of the writer that through this great extent of Huronian formation in the Rainy River and Lake of the Woods district, out of the many veins which have already been located (the number of which locations will be immensely increased in the next few years), as large a proportion of paying gold-mines will be developed as are found as an average in successful mining districts.

Conveniences of the District.—The district enjoys very fair transportation-conveniences, which are being constantly improved.

Wages are low in comparison to the west, miners being paid from $1.50 to $2.50 per day.

As an example of the cost of mining in the Lake of the Woods:—a shaft, 6 feet by 16 feet, cost $35 per foot; winzes, 5 feet by 7 feet, $25 per foot; drifts, 5 feet by 7 feet, $15 per foot; stoping on a 3- to 4-foot vein, $3 per ton; milling in a 10-stamp mill, about $1.75 per ton.

On the Seine river, sinking costs from $25 to $40 per foot, drifting by machines about $10 and hand-drifting $15 per foot in the granite.

Output.—The gold-yield of the Province of Ontario has been almost entirely from the Rainy River district. The Bureau of Mines gives the following returns of output:

For the year ending October 31, 1893, . .	$32,960
" " " 1894, . .	32,776
" " " 1895, . .	50,281
" " " 1896, . .	121,848

Examples of Country-Rocks.—The following detailed descriptions of specimens of country-rocks, as determined by microscopic examination of prepared sections, may be of interest. The photographs of the slides unfortunately did not bring out

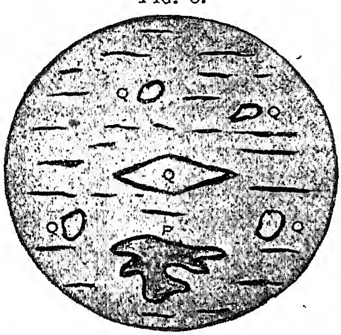
the different minerals distinctly, and therefore somewhat crude sketches have been reproduced here.

My thanks are due to Prof. Miller, of the Kingston School of Mining, for assistance in connection with the rock-sections.

1. A schist (Fig. 5) forming the country-rock at Bath Island, Lake of the Woods, consists of masses of quartz and iron pyrites with a schistose ground-mass, principally of quartz, mixed with untwinned feldspar and a very few streaks of chlorite. The rock has been subjected to great pressure parallel to the vein.

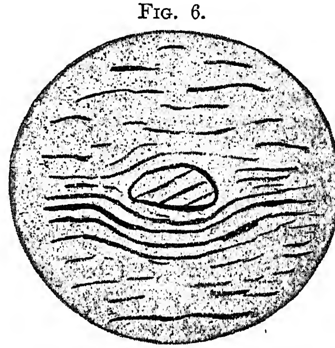
2. A schist, much decomposed, consisting of a fine-grained feldspar, with chloritic masses, a little unaltered pyroxene, some calcite and a number of small specks of pyrite, forms the coun-

Fig. 5.



Schist at Bath Island. Q, quartz; P, pyrite.

Fig. 6.



Schist at Arrastra Vein, near Rat Portage.

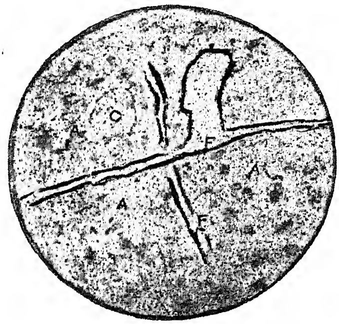
try-rock of the Lyall mine in the Rainy Lake, the operation of which has been unsuccessful hitherto.

3. A schist (Fig. 6) forming the country-rock at the Arrastra vein, east of Rat Portage, Lake of the Woods, consists of a crushed schistose mass. The chief matrix is feldspar, with some quartz and mica. Streaks or lines of the latter show flow-structure around masses of the original rock from which the schist has been formed. One of these masses showed a bedding, or schistose structure, at an angle of 30° to that of the present structure.

4. A greenstone (Fig. 7) forming the country-rock of Pine Portage mine, Lake of the Woods, is much crushed, and consists chiefly of a crushed matrix of chlorite (resulting from the

decomposition of pyroxene), some feldspar and pyroxene. The feldspar is present chiefly as little veinlets. The rock is an altered diabase, which would probably be found at greater depth in its unchanged condition.

FIG. 7.



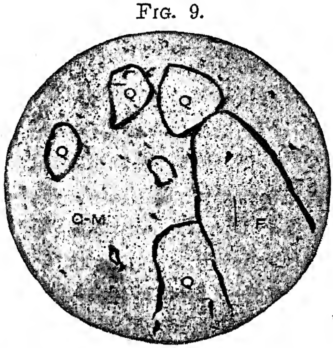
Country-Rock, Pine Portage Mine.
A, pyroxene; C, chlorite;
F, feldspar.

FIG. 8.

Country-Rock, Sultana Mine.
F, feldspar; H, hornblende;
P, pyrite.

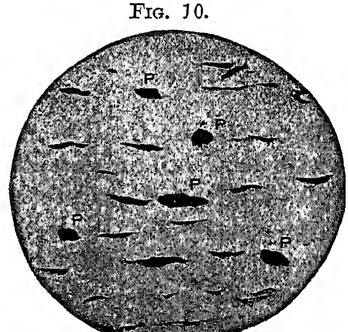
5. The country-rock of the Sultana mine (Fig. 8) shows signs of crushing and alteration. Feldspar is the predominant mineral, both twinned and untwinned with quartz, mica and a little

FIG. 9.



Altered Granite or Protogene, Seine River. Q, quartz; F, feldspar; G-M, ground-mass (talc, etc.).

FIG. 10.



Felsitic Schist from Garnier's Island. P, pyrite.

hornblende and pyrite. Where alteration is most apparent the feldspar has been changed into mica, and commences to show a schistose structure. The rock may be a granite (or the "grani-

tite" of G. Rose) which has been squeezed by a greenstone upheaval alongside.

The adjacent greenstone mass, which is much decomposed near the surface, is made up for the most part of secondary hornblende (uralite) and feldspar (twinned and untwinned), with calcite, pyrite and magnetite as accessories. At greater depth it may become evident that the mass is a diabase dike.

6. An altered granite or protogene (Fig. 9), consisting of masses of quartz, feldspar (chiefly orthoclase) and a greenish ground-mass of talcose and chloritic composition, is representative of the areas of altered granite which are found to carry a great number of veins in the Seine River district.

7. A felsitic schist from Garnier's Island, Lake of the Woods (Fig. 10), consists of fine-grained feldspar for the most part, some pyrite crystals and a very little mica. This schist changes into a massive felsite dike, consisting of massive fine-grained feldspar, a good deal of pyrite in little masses scattered through it, and a very little mica.

The Microstructure of Steel and the Current Theories of Hardening.

BY ALBERT SAUVEUR, CHICAGO, ILL.

(Colorado Meeting. September, 1896.)

INTRODUCTION.

To understand this paper properly, the reader must have a clear conception of what is meant by the "critical points" of iron and steel;* and in order to avoid the necessity of tedious references to previous publications on the part of those unfamiliar with the subject, it is here briefly recapitulated.

Critical Points.—If a piece of steel containing, say, 0.50 per cent. or more of carbon, be heated to a high temperature and then allowed to cool slowly, the cooling proceeds at first at a uniformly retarded rate until a temperature of about 700° C. is reached, when there is a sudden "retardation" in the fall of

* See H. M. Howe's "Heat Treatment of Steel," *Trans.*, xxiii., 466.

the temperature, indicating an evolution of heat, often so considerable as to cause a momentary stop in the cooling, or even an actual rise of the sensible temperature,—a "recalescence" of the cooling metal. After this retardation the fall of temperature resumes its normal rate, which is continued until atmospheric temperature is reached. This retardation during the cooling, which indicates that some important change, evolving heat, is taking place within the metal, is called a "critical point." Mr. Osmond, who was the first to determine accurately, by means of the excellent pyrometer of Mr. Le Chatelier, the position and magnitude of the critical points, adopted Chernoff's notation, and designates the critical points by the letter A.

In heating, as would be expected, there is a reverse phenomenon, an absorption of heat causing a retardation in the rise of temperature. To distinguish the critical point which occurs in cooling from that which occurs in heating, the former is called A*r* the latter A*c*. The two retardations, however, do not take place at the same temperature, the critical point A*c* being situated some 30° higher than A*r*.

Mr. Howe has shown conclusively that, in order to induce the retardation A*r*, or rather the change which such retardation implies, the steel must be first heated past the point A*c*; and reciprocally, the change which occurs at A*c* cannot take place unless the steel has first been cooled to a point below A*r*. Clearly, therefore, the retardations A*r* and A*c*, although not taking place at exactly the same temperature, are simply the opposite phases of the same phenomenon. The change at A*r* is the reversal of that at A*c*.

Howe and Osmond have shown that by hastening the cooling the critical point A*r* is proportionally lowered until, when the cooling is sufficiently rapid, as in quenching, there is no retardation; the change A*r* does not take place. Hence certain conditions which existed above A*r* are retained by quenching; and to this retention the various theories attribute the hardness of quenched steel. It is only in regard to the nature of what is thus preserved by sudden cooling that they differ. The heat of the retardation A*r* remains latent in hardened steel, and can be made apparent, according to Mr. Osmond, by dissolving the metal in double chloride of ammonium and copper, when it evolves more heat than unhardened steel. This latent heat is

also liberated during tempering and produces an acceleration in the rate of heating.* It is reasonable to suppose that if we could heat our metal as suddenly as we can cool it, it would be possible to raise its temperature past the point *Ac*, without inducing the *Ac* change, *i.e.*, the change which on subsequent sudden cooling produces hardness, so that if the steel were then quenched it would not be hardened.

High-carbon steels and those of medium hardness have only one critical point. Such is not the case, however, with softer steels, which exhibit, on cooling or heating, two or three critical points. To distinguish these, they are called respectively Ar_3, Ar_2, Ar_1, and similarly, in heating, Ac_3, Ac_2, Ac_1.

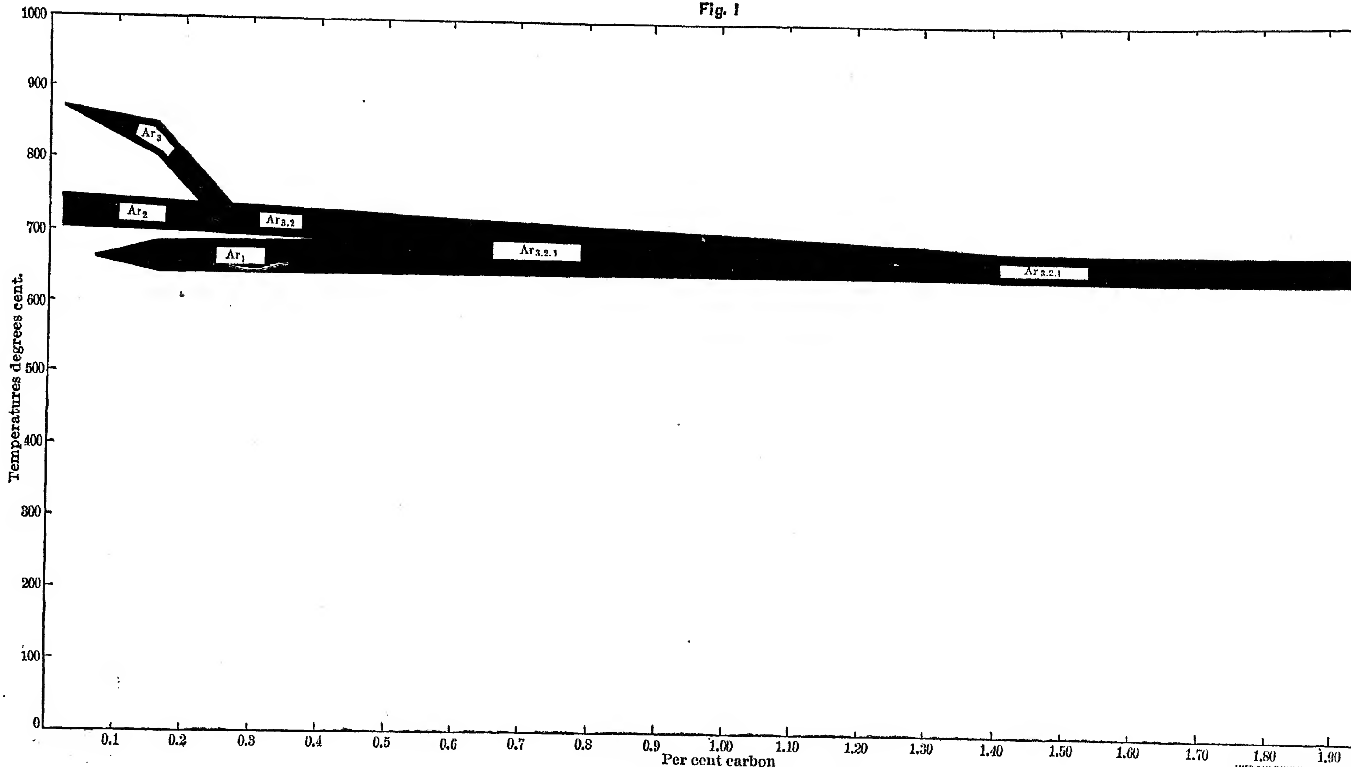
It has been argued that at about 0.25 per cent. of carbon the two upper retardations run together, so that Ar_2 is no longer a single point, but in reality a double critical point, which should be called $Ar_{3,2}$. Similarly, above 0.50 per cent. of carbon, instead of a single critical point, we have a triple point $Ar_{3,2,1}$. This notation implies that the three changes which occur separately in soft steel take place simultaneously in high-carbon steel at $Ar_{3,2,1}$: while from the notation Ar_1 it would be inferred that we have here only one change, the two upper transformations of mild steel being absent. It is generally accepted that the single evolution of heat exhibited by the cooling of high-carbon steel is caused by three distinct changes occurring simultaneously; but this point has never been proved satisfactorily, and the notation which implies it is here adopted for convenience, and with this understanding. The retardations, considered collectively, will be hereafter called the *critical range*, regardless of the number of critical points contained.

Fig. 1 shows graphically the position of the critical points in *cooling* steels of various carbons. The width of the black lines does not refer to the intensity of the retardations, but only indicates the range of temperature which they cover. For instance, it shows that the single retardation of high-carbon steel begins at about 680° C. and ends at about 640° C. The maximum evolution of heat lies somewhere between these limits, but not necessarily in the middle.

This graphical representation was obtained by plotting the re-

* "On the Critical Points of Iron and Steel," F. Osmond. *Jour. Iron and Steel Inst.*, No. 1, 1890, p. 38.

Fig. 1



POSITION OF THE CRITICAL POINTS

sults of the investigations of Osmond, Howe, Roberts-Austen, Arnold and the writer; and, with one or two exceptions, all their figures fall very nearly within the limits here indicated.

We see that between 0.02 and 0.07 per cent. of carbon, steels have only the two upper retardations, Ar_3 and Ar_2, while between, say, 0.07 and 0.22 per cent. of carbon, we find three distinct critical points, Ar_3, Ar_2, Ar_1. Again, between 0.22 and about 0.43 per cent. of carbon there are two retardations, $Ar_{3,2}$, and Ar_1, the upper one being absent. Above 0.43 per cent. of carbon, as already stated, there is only one critical point, Ar_1.

Like the single retardation of hard steel, the multiple retardations of soft steel indicate reversible changes. Ac_3 and Ac_1 occur at a temperature somewhat higher than Ar_3 and Ar_1, while Ac_2 and Ar_2, on the contrary, take place at nearly the same temperature. Certain impurities, especially manganese, nickel, chromium and tungsten, lower the critical points very decidedly. Manganese and nickel, when present in sufficient quantity, apparently eliminate them (probably by lowering them below the atmospheric temperature). It is also probable, roughly speaking, that the purer the steel, the longer will the three critical points remain separated, as the percentage of carbon increases.

The meaning attributed to the retardations will be discussed subsequently. It is only necessary for the present to bear in mind what is meant by "critical points." They are those retardations which occur during the heating and cooling of a piece of steel, and which indicate that some important changes, absorbing or evolving heat, are in progress; which changes, as will be seen, greatly affect the chemical, physical and mechanical properties of the metal.

Purpose of this Paper.—In the present paper it is proposed, first, to describe at some length the changes of microstructure which occur, during slow cooling, in steels containing various amounts of carbon, and secondly, to examine what bearing, if any, such structural changes as occur at the critical points, have upon the current theories of hardening.

The paper has accordingly been divided into two parts: I. The Microstructure of Steel, and II. The Current Theories of Hardening.

I.—The Microstructure of Steel.

Development of the Microstructure of Polished Sections.—It will not be out of place here to say a few words with regard to the development necessary to bring out the microstructure of polished sections of iron and steel, as this is a point of vital importance in the microscopy of iron.

If a polished surface of steel be examined through the microscope, unless it contains constituents of very different hardness, it will reveal little or nothing of the structure. If examined by oblique reflected light, the bright surface, acting like a mirror, will reflect the totality of the light outside the objective and the specimen will appear uniformly dark. If direct reflected light be used, all the rays will be reflected back into the objective, and the object will appear equally bright all over the field.

It is therefore necessary to differentiate the various constituents in such a way as to make them distinguishable under the microscope. This may well be called a development of the structure. It is usually accomplished by means of some chemical action, an etching-process, which colors differently some of the constituents, while others are left unaltered and brilliant.

A defective etching is most unsatisfactory; it distorts the true structure of the object, giving it an appearance very liable to lead to erroneous deductions. A properly conducted development, on the contrary, seldom fails to bring out the structure clear and brilliant, and is then always a delightful and convincing revelation of the great possibilities of the microscope in the domain of the physics of steel. The writer soon learned to discard, on account of its unreliableness and many other objections, the etching with very dilute acid, which has been so universally used, and to adopt the following method:

The polished sample, suitably held, is dipped into concentrated nitric acid (1.42 sp. gr.) which, on account of the passivity of iron, has little or no action on the polished surface. The specimen is then placed under an abundant stream of running water, and the acid is quickly and completely washed off. As soon as the layer of concentrated acid which covers the surface is diluted by the running water, it attacks the steel, at first vigorously, but for such a very short time (since

the water soon removes all trace of acid) that there is no danger of etching too deeply. Such treatment develops the structure sharply and clearly, the etching being of uniform intensity all over the surface, and free from the objectionable colored film and from the unlike appearance of different parts of the field, caused by local actions of varying intensity, which are so troublesome and misleading in etchings with dilute acid. It is sometimes necessary to repeat the treatment, in order to develop the structure to the proper depth; but more than two immersions are seldom required. The specimen is then washed in alcohol and quickly dried with a soft cloth, or, better still, under a blast-jet, if such is at hand.

Until the recent publication of Mr. Osmond's remarkable paper on "Microscopic Metallography,* this had been the process exclusively used by the writer. In that paper, Mr. Osmond describes a method of etching with tincture of iodine, which gives excellent results. The *modus operandi* is as follows: Upon the polished surface pour a drop or two (Mr. Osmond recommends one drop per square centimeter) of ordinary tincture of iodine, and let the solution act until it is discolored. Wash in alcohol and dry quickly with a soft cloth or under a blast-jet. The writer prefers to use a solution diluted with alcohol, one volume of tincture to one of alcohol. Two or more applications may sometimes be required before the specimen is sufficiently etched. Both methods are good; and it is difficult to say when one should be used in preference to the other. It is probable that to bring out the finest details of the structure, such as the delicate laminæ of pearlyte, the treatment with iodine is preferable; while for the examinations with low power the nitric acid method is more effective. But even this rule will not always hold true.

There is another method of finishing the preparation, of which a few words should be said. It may be called "polishing in relief," and consists in finishing the polishing upon a yielding support, so that the various constituents will be worn unequally, the hardest standing in relief. When the components differ much in hardness, the microscope will reveal the

* *Méthode générale pour l'Analyse Micrographique des Aciers au Carbone.—Société d'Encouragement pour l'Industrie Nationale.* Paris, Mai, 1895.

structure without further treatment. This method affords a good means of judging the relative hardness of some constituents. A good support for such polishing consists of a block of soft wood, upon which is stretched a piece of parchment, thoroughly soaked with water and covered with a very little of the finest jewellers' rouge.

Microscopic Constituents of Steel.—Let us first examine the general nature and properties of the various microscopic constituents of steel; and later we will discuss their mode of occurrence in steel containing various amounts of carbon, and quenched at different temperatures.

All carbon-steels, whether quenched or slowly cooled, are made up of one or more of four primary constituents, which have been called respectively: *ferrite*, *cementite*, *pearlyte*, and *martensite*. The three first names were proposed by Mr. Howe, the last by Mr. Osmond. Prof. Arnold has recently sounded a cry of warning against the danger of giving ill-fitted names to these constituents, or the same name to different constituents. The writer fully appreciates the importance of well-chosen names, and the confusion which must result from the use of synonyms and of homonyms; and this is precisely why he does not understand the reluctance or hesitation of Prof. Arnold and a few other steel microscopists in adopting Mr. Howe's names of ferrite, cementite and pearlyte for the components of unhardened steel. Nobody will question the propriety of giving mineralogical names to the microscopic constituents of metals, considering the analogy which exists between petrography and metallography; and it is doubtful whether better names than these could be found, so acceptable alike to English, French and German scientists; for, in each language, these terms are in some measure suggestive of the nature or the appearance of the constituents. There would be very little danger of confusing the student, if the metallographists who still hold aloof would gracefully adopt this nomenclature, which, besides its other claims, has also the right of priority; for while Dr. Sorby was the first to study and describe the constituents of steel, Mr. Howe was the first to christen them.

Ferrite.—This is iron free from carbon. Whether it is free also from the other impurities usually present in steel has never been conclusively shown. Indeed, it is quite probable that the

grains of ferrite are sometimes impregnated with impurities. Ferrite presents three principal modes of occurrence.

1. When present in considerable quantity, as in soft steel slowly cooled or quenched below the critical range, it segregates in relatively large masses, and crystallizes in polyhedric grains, which are interfering crystals of the isometric system, mostly cubes and octohedra (see Plate I., Figs. 2, 3, 4, and Plate III., Fig. 4). In the drawings the ferrite is left white, the black net-work showing the outlines of the grains.

2. In steel of medium hardness, which has been slowly cooled from a high temperature, the ferrite, present only in small quantity, forms a membrane of varying thickness enveloping the grains of pearlyte. A section cut at any angle will therefore exhibit a net-work structure, the pearlyte forming the meshes (see Plate III., Fig. 6). If such steel, however, be forged until a relatively low temperature is reached, or if it be reheated to a low temperature (a little above the critical range), the pearlyte and ferrite are more thoroughly mixed in a somewhat confused manner, presenting an amorphous appearance.

3. Ferrite forms a structural element of pearlyte, which, as will be seen presently, is made up of an intimate mixture of ferrite and cementite. To distinguish it from this variety, the ferrite, which in the structure exists by itself in segregated masses, will here be called structurally free, or segregated ferrite.

Structurally free ferrite forms, of course, the totality of carbonless iron. As the carbon-content increases, the free ferrite diminishes, until at about 0.80 per cent. of carbon it entirely disappears, the steel being then wholly made up of pearlyte (see Fig. 4 and Plate I., Fig. 12).

The etching of the polished sections with iodine or nitric acid leaves the ferrite white and brilliant, and develops the joints between the grains (see Plate III., Fig. 4). On etching more deeply, however, some of the grains are slightly colored or take a mottled appearance, while others remain bright. Ferrite being the softest of all the constituents in the field, it is more easily and quickly worn away by polishing, so that under the microscope all the other constituents stand in relief with regard to it; this constitutes an additional means of identifying it.

Cementite.—This is iron combined with cement-carbon, *i.e.*, carbon as it exists in unhardened steel. It is a carbide, answering to the formula Fe_3C. As early as 1885, Abel and Müller, working independently, ascertained the presence in unhardened steel of such a carbide; and the more recent researches of Arnold and Read* have fully confirmed their conclusions, so that there is now hardly any possible doubt concerning the chemical composition of cementite. In steel containing much manganese, however, it is very probable that this constituent is a double carbide of iron and manganese.

Like ferrite, cementite occurs:

1. In segregated masses in very hard steel, but it always remains structureless (see Plate III., Fig. 5). The white constituent standing in relief is cementite, the dark is pearlyte. In the drawings of Plate I. the cementite is shown in black, to distinguish it from the ferrite, which is left white.

2. As a very thin membrane, around the grains of pearlyte, when present in small quantities and under certain conditions (see Plate I., Fig. 14).

3. As a structural element of pearlyte.

Cementite remains bright and brilliant, even after repeated etchings of iodine or nitric acid. It has a more metallic luster than ferrite; and, to an experienced eye, this appearance alone will suffice to identify it. While ferrite is the softest, cementite is the hardest of the constituents, so that it stands in relief relatively to the others, especially if the polishing is finished on a soft, yielding support. Moreover, free ferrite has a granular structure, while cementite is always structureless. Again, cementite cannot be scratched by a needle, while the soft ferrite is easily marked in this way. Structurally free cementite is not found in soft or medium-hard steel. It begins to appear in steel containing about 0.90 per cent. of carbon, and then increases in quantity proportionally to the carbon-content. In steel of 2 per cent. carbon it forms about 23 per cent. of the mass (see Fig. 4 and Plate I., Fig. 16). Structurally free ferrite and structurally free cementite never exist together in the same steel. The former is the associate of pearlyte in soft and

* "The Chemical Relations of Carbon and Iron," F. O. Arnold and A. A. Read, *Jour. Chem. Soc.*, lxv., 788, August, 1894.

medium-hard steel, the latter the associate of pearlite in highly carburetted steel (see Fig 4).

Pearlyte.—As already stated, pearlyte is not, strictly speaking, an elementary constituent of steel; but it is, nevertheless, a very distinct one, and should be treated as such rather than as a mere mixture of ferrite and cementite.

A magnification of at least 300 diameters is required to resolve satisfactorily the structure of pearlyte. We find then, as first pointed out by Mr. Osmond, that it is necessary to distinguish between lamellar and granular pearlyte. The lamellar variety is best found in steel very slowly cooled from a high temperature, and especially in annealed steel. The longer the annealing, the more pronounced the character of the structure. It is called lamellar, because it is made up of very thin plates or lamellæ, alternately of ferrite and cementite. This structure gives rise, under the microscope, to a beautiful play of color, strongly suggestive of mother-of-pearl. Dr. Sorby, who first discovered it, called it the pearly constituent, and Mr. Howe pearlyte.

The presence of well-defined lamellar pearlyte is certain proof that the steel has been annealed or cooled slowly and undisturbedly from a high temperature. The granular variety is found in steels that have been forged down to a comparatively low temperature, or which have been reheated to a low temperature. As the name indicates, it is somewhat granular in appearance, consisting of a mixture of small irregular grains of cementite and ferrite. Pearlyte is always present in unhardened steel; at first in a very small quantity, but increasing rapidly with the carbon-content. When the steel contains about 0.80 per cent. of carbon, the whole mass consists of pearlyte. With further increase of carbon the amount of pearlyte diminishes, being gradually replaced by structurally free cementite (see Fig. 4).

Pearlyte is colored dark both by iodine and by nitric acid. It always forms the dark constituent of unhardened steel.

As just pointed out, in steel with about 0.80 per cent. of carbon, pearlyte is the sole constituent. Prof. Arnold very properly calls this point "the saturation point" of steel; and he finds that in steel containing very little impurity besides carbon, about 0.90 per cent. of carbon is required for satu-

ration, *i.e.*, that not until then does all the structurally free ferrite disappear. With commercial steels, however, such as those used in connection with the present investigation, containing from 0.40 to 1.00 per cent. of manganese, and from 0.75 to 1.25 per cent. of total impurities, the saturation-point is very near 0.80 per cent. of carbon.

It is quite certain that the purer the steel the more carbon is required to saturate it. This, no doubt, is partly due to the fact that an impure steel contains less iron to be saturated, and also to the presence of considerable manganese, which, forming a double carbide, helps to saturate, further reducing thereby the necessary amount of carbon. The pearlyte of commercial steel therefore contains about 0.80 per cent. of carbon, or about 12 per cent. of cementite (Fe_3C), and 88 of ferrite; roughly, 1 part of cementite to 7 parts of ferrite.

The structure of unhardened steel is explained as follows: The carbon present in the steel unites with a portion of the iron to form Fe_3C or cementite. The ferrite and cementite then unite structurally in definite proportions to form pearlyte, leaving, as the case may be, an excess either of ferrite or of cementite, the former in soft steel, the latter in highly carburetted steel, as shown in Fig. 4. Knowing the compositions of cementite and pearlyte, the theoretical microstructural composition of any carbon-steel can easily be calculated. If, for instance, the steel contains 0.50 per cent. of carbon, we know that this amount of carbon will unite with a portion of the iron to form cementite, or Fe_3C (6.67 C + 93.33 Fe) in the following proportion:

$$\overset{\text{C.}}{6.67} : \overset{\text{Fe}_3\text{C.}}{100} = \overset{\text{C.}}{0.50} : x,$$

yielding therefore 7.50 per cent. cementite. This cementite unites structurally with an additional amount of iron, forming pearlyte (12 Fe_3C + 88 Fe), as follows:

$$\overset{\text{Fe}_3\text{C.}}{12} : \overset{\text{Pearl.}}{100} = \overset{\text{Fe}_3\text{C.}}{7.50} : x,$$

or 62.50 per cent. of pearlyte. The balance of the steel, or 37.50 per cent., is made up of structurally free ferrite. If the steel contains more than 0.80 per cent. (say 1.20, for instance) of carbon, we have an excess of cementite. The composition

can then be determined as follows: 1.20 per cent. of carbon yields 18 per cent. of cementite. This leaves 82 per cent. of free iron, which will unite with part of the cementite to form pearlyte, as follows:

$$\begin{array}{ccccc} \text{Fe.} & & \text{Fe}_3\text{C.} & & \text{Fe.} \\ 88 & : & 12 = & 82 : & x, \end{array}$$

or 11.18 per cent. of cementite, leaving therefore 6.82 per cent. of structurally free cementite. Hence the structure of such steel is made up of 93.18 per cent. of pearlyte and 6.82 per cent. of structurally free cementite.

These figures would probably be absolutely correct, if the steel contained nothing but iron and carbon. The impurities always present in commercial steel, however, will necessarily alter them—which renders these calculations only approximate. In the following table, the microstructural composition of unhardened steel, containing various amounts of carbon, has been calculated in this way, the decimals being omitted.

TABLE I.—*Theoretical Microstructural Composition of Unhardened Carbon-Steels.*

Carbon. Per Cent.	Microstructural Composition. Per Cent.		
	Pearlyte.	Ferrite.	Cementite.
0.	0	100	0
0.10	12	88	0
0.20	25	75	0
0.30	37	63	0
0.40	50	50	0
0.50	62	38	0
0.60	75	25	0
0.70	87	13	0
0.80	100	0	0
0.90	98	0	2
1.00	97	0	3
1.10	95	0	5
1.20	93	0	7
1.30	91	0	9
1.40	90	0	10
1.50	88	0	12
1.60	86	0	14
1.70	85	0	15
1.80	83	0	17
1.90	81	0	19
2.00	80	0	20
2.10	78	0	22
2.20	76	0	24
2.30	74	0	26
2.40	73	0	27
2.50	71	0	29

Martensite.—Martensite is that constituent which exists at a high temperature, and being retained by sudden cooling confers hardness upon quenched steel. While we know with reasonable certainty the composition of ferrite, cementite and pearlyte, we know very little about the true nature of martensite. Indeed, if we could discover its composition, the hardening of steel would no longer be an unsolved problem. Each theory which has been advanced to explain the hardening of steel has attributed to martensite the composition demanded by that theory. Of its real composition, however, we have no direct evidence.

Martensite contains iron and carbon, but not in definite proportions like pearlyte; the amount of carbon it contains varying from 0.12 per cent. in very soft steel, quenched above the critical range, to 0.90 per cent. in hard steel. It forms about 75 per cent. of very mild steel quenched (the balance being ferrite); while it constitutes the whole bulk of hardened steel containing from 0.25 to 0.80 per cent. of carbon. In more highly carburetted steel some structurally free cementite is found besides the martensite (see Fig. 4 and Plate I.). As might be expected, therefore, the hardness of martensite varies greatly. In its most dilute form (when it contains the greatest proportion of iron) it is relatively soft, while when highly carburetted, it becomes intensely hard, although never as hard as cementite.

The composition of martensite will be discussed at greater length a little farther on.

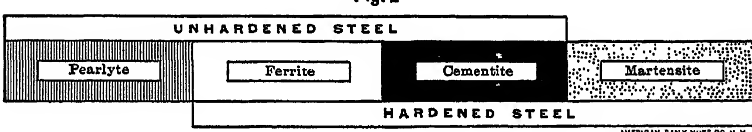
Martensite is colored both by iodine and by nitric acid; but the intensity of its coloration varies with the degree of carburization. When much diluted, as in very soft steel quenched above the critical range (Plate I., Fig. 1), it assumes a light yellowish tint, which increases in depth with the increase of carbon. Highly carburetted martensite (*i.e.*, the martensite of high-carbon steel) is colored much darker by the etching—as dark and even darker than pearlyte under similar treatment. Dilute martensite is easily distinguished from pearlyte by its light yellow color. With the more carburetted or concentrated variety, the color alone is no longer a sure guide, and we must then differentiate them by their structure. If the pearlyte is lamellar, no confusion is possible, for its peculiar structure will

readily identify it. With granular pearlyte the differentiation is more difficult, and the structure must then be closely examined. Martensite, as Mr. Osmond has remarked, often presents groups of parallel needles, or rectilinear fibers, which cross each other frequently.

It should be remembered that very high power is necessary to resolve the structure of martensite, and even then the details are so minute and often so confused that what is revealed teaches us little with regard to its nature. The writer finds strong indications, however, that martensite, like pearlyte, is made up of two components differing in hardness.

To sum up, all unhardened steels are composed of pearlyte alone, or pearlyte associated either with ferrite or cementite. All hardened steels are composed of martensite alone, or martensite associated with ferrite or cementite. This is shown graphically in Fig. 2.

Fig. 2



Mr. Osmond, in his paper already referred to, describes two additional constituents, which he calls, respectively, sorbite and troostite. They have not been considered here, because the writer has not been able to identify them satisfactorily, and also because they are, at best, "transition-forms," as Mr. Osmond calls them, having but a short existence in the structural changes induced in steel by heat-treatment. Even were their existence proved beyond a doubt, it would not in any way affect our deductions based on microscopical investigations. Mr. Osmond finds that on etching with iodine a high-carbon steel reheated to a little above the critical range, part of the pearlyte is either entirely colored, or only as to some of its lamellæ, and as both ferrite and cementite should remain bright after such a treatment, he infers that we are here in the presence of a new constituent, *sorbite*, and that the pearlyte of such steel is made up of pearlyte and sorbite, or cementite and sorbite, or even of two sorbites differently colored. He supposes that it contains some hardening carbon, and that it represents a *zone of transition*

between the ferrite and cementite in unhardened steel, and between cementite and martensite in quenched steel.

The constituent *troostite* Mr. Osmond finds in steel quenched during the critical points, between the martensite and the ferrite. It is a "transition-form" between ferrite and martensite. Nothing is said regarding its chemical composition.

Microstructure of Steel Quenched Above, During and Below the Critical Range.—In studying the microstructure of quenched steel, the writer has had more especially in view to ascertain whether there were any perceptible changes corresponding to the critical points, and, if any, to determine their nature. The simplest way of accomplishing this purpose seemed to be to heat considerably above the critical range a number of steel bars containing different amounts of carbon, and to quench them, after slow cooling, at various temperatures above, between and below the critical points. In this way we retain in the cold metal the structural arrangement which prevailed at the quenching-temperature, and we are thus enabled to judge of the changes of structure occurring in steel during slow cooling, and to ascertain what relations they bear to the critical points.

Six samples of steel were accordingly selected, containing respectively 0.09, 0.21, 0.35, 0.80, 1.20 and 2.50 per cent. of carbon. The first three samples were kindly furnished by Mr. Howe, who had accurately determined the position of their critical points; the last three samples were treated by the writer. They were all commercial steels, containing from 0.40 to 0.90 per cent. of manganese, and from 0.60 to 1.00 per cent. of total impurities (carbon not included).

Several bars of each series were heated well past the critical range and allowed to cool partially before quenching, one bar being quenched above Ar_3, another between Ar_3 and Ar_2, the next between Ar_2 and Ar_1, and the last below Ar_1 (see Fig. 1).

The structures of these various samples were drawn from the microscope, being uniformly magnified 250 diameters and arranged comprehensively in Plate I. In steels which do not possess the upper retardation Ar_3, no change of structure takes place above Ar_2, and in steels which have only the lower retardation Ar_1 there is only one change of structure. This explains

the arrangement of the drawings. The areas occupied by the various constituents were measured with the planimeter, and their percentages are tabulated in Table II.

Planimeter Measurements.—A short description of the manner in which these measurements are conducted may be of interest to those engaged in similar investigations. The results shown in Table II. were obtained from areas at least twice as large as

TABLE II.—*Microstructural Composition of Some Quenched Carbon-Steels.*

Steel No.	Carbon. Per Cent.	Quenched Above Ar_3. Per Cent. in Vol.*			Quenched Between Ar_3 and Ar_2. Per Cent. in Vol.			Quenched Between Ar_2 and Ar_1. Per Cent. in Vol.			Quenched Below Ar_1 or Slowly Cooled. Per Cent in Vol.		
		Mart.	Fer.	Cem.	Mart.	Fer.	Cem.	Mart.	Fer.	Cem.	Pearl	Fer.	Cem.
1	0.09	77	23	0	27	·73	0	11	89	0	10	90	0
		Quenched Above Ar_2. Per Cent. in Volume.											
		Martensite.		Ferrite.		Cementite.							
2	0.21	100		0		0		31	69	0	23	77	0
3	0.35	100		0		0		56	44	0.	50	50	0
		Quenched Above Ar_1. Per Cent. in Volume.											
		Martensite.			Ferrite.			Cementite.					
4	0.80	100			0			0			100	0	0
5	1.20	94			0			6			92	0	8
6	2.50	80			0			20			77	0	23

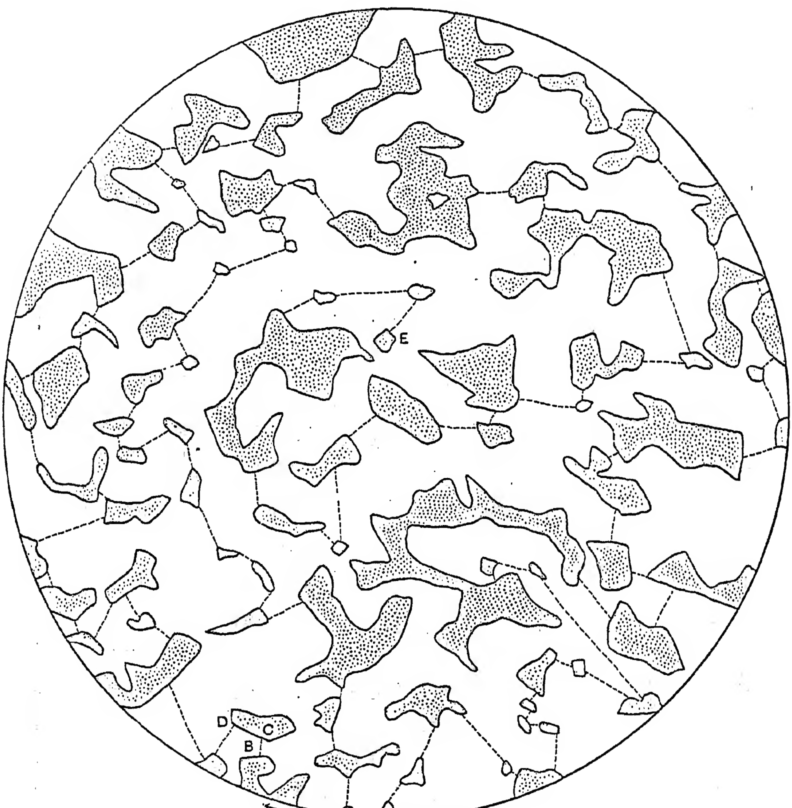
those reproduced in Plate I. As an illustration, let us take the bar containing 0.09 per cent. carbon and quenched between Ar_3 and Ar_2. The actual field measured is shown in Fig. 3. For the sake of simplicity, the joints of the grains of ferrite are not shown in the drawing.

It is proposed to ascertain the total area occupied by the martensite. If we were to measure each little area indepen-

* If a body containing a great many particles is cut by a plane, a certain number of the particles will be intersected, and the ratio of the sum of these small a·eas to the total area of the section is practically equal to the ratio of the volume of the small particles to the total volume of the body. Mathematically, however, the two ratios are equal only if the number of particles is infinite and the uniformity of distribution perfect.

dently, the work would be very long and tedious; and, besides, the great number of readings and adjustments of the planimeter required would introduce a serious source of error. To simplify the operation and greatly reduce the possible error, all the small areas are united in a continuous chain, as shown

FIG. 3.



Section of 0.09 = Carbon-Steel, quenched between Ar_3 and Ar_2.

by the dotted lines, in such a way that starting from the point A, and passing from one grain to the other, we finally reach the point E, thus coming in contact with all the small areas in our travel. We then proceed as follows: The planimeter set at zero is placed at A, and we start, moving it in the direction indicated by the arrow, as if to measure the small area; when

we reach the point B, however, we pass over to the next grain, following along the dotted line BC; from the point C we pass to D, following the outline of that grain; from D to the next area; and so on until we reach the point E on the last small area. We then follow the whole outline of this grain, which brings us back to E, and from E we proceed on our return trip, again passing from one grain to the next in the reverse order, but this time going over that portion of the outline of each grain which we had previously not travelled over. It is evident that by proceeding in this way we have followed with the planimeter the complete outline of each grain of martensite. The error introduced by going over the dotted lines in one direction is exactly neutralized by going over them in the opposite direction on our return-journey, and the reading of the planimeter gives the total area of the martensite. This method is quick, requiring but one adjustment of the planimeter and one reading; it is accurate, there being no trouble in checking the results.

The structural compositions of the various samples were ascertained in this way, and the results shown in Table II. were plotted in Fig. 4, which gives a graphical representation of the microstructural composition of any carbon-steel quenched either above, between or below the critical points, as well as when slowly cooled, and will assist us in understanding the nature and magnitude of the structural changes which occur at those points.

As already mentioned the amount of impurities (besides carbon) have some influence upon the relative proportions of the microscopic constituents. In very pure steel, for instance, the saturation-point is near 0.90 per cent.; so that we cannot from Fig. 4 infer the composition of any carbon-steel with absolute accuracy; but it gives approximate results, quite satisfactory in the case of commercial steel.

The writer has repeatedly checked by planimeter-measurement the carbon per cent. found by chemical analysis, on the assumption that pearlyte contains 0.80 per cent. of carbon. To obtain comparative results, however, the steel should always be heated to the same temperature (preferably just above the critical range) previous to the microscopical examination, as it has been found that heating to different temperatures somewhat alters the volumes occupied by the pearlyte.

From the drawings of Plate I. and the figures of Table II., we see that *each critical point is accompanied by a marked structural change.* It will be shown subsequently that the changes of structure correspond *exactly* with the retardations, beginning and ending with them.

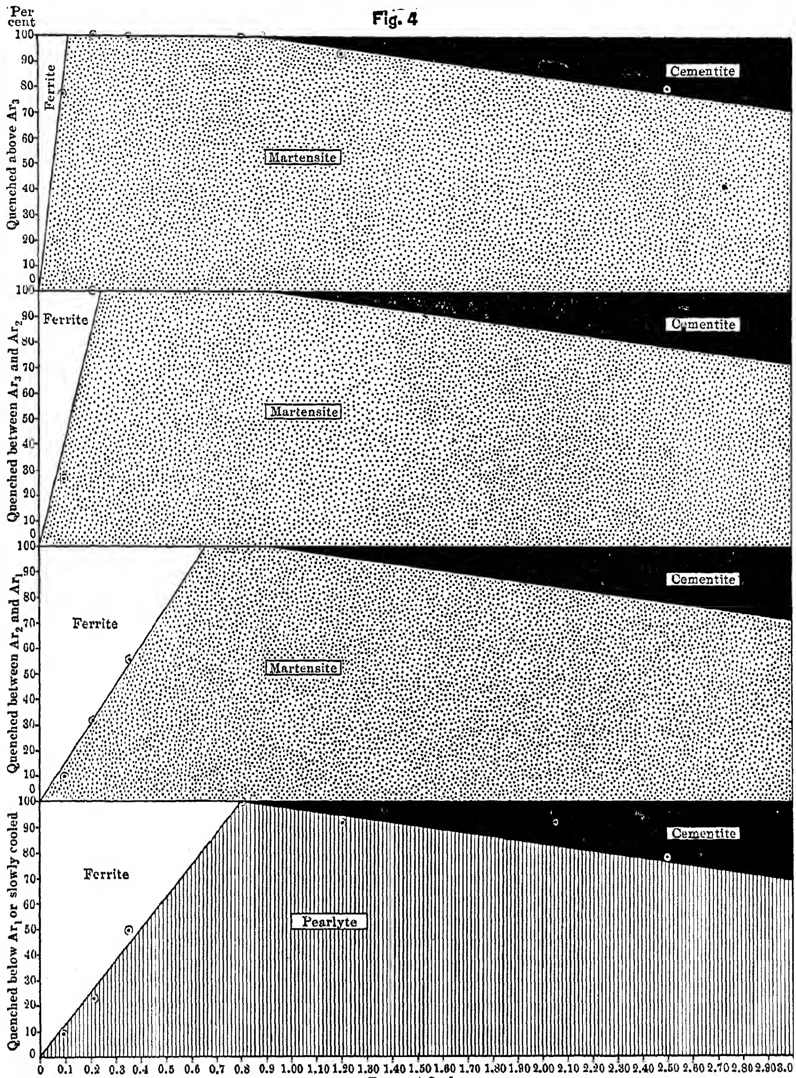
Let us now examine separately the structural changes which occur in each series of steel here investigated.

Steel No. 1—Carbon 0.09 *Per Cent.*—The structural changes which occur in steel at the critical points are so considerable, even in the case of a soft steel containing only 0.09 per cent. of carbon, that they have been further illustrated in Plate III., Figs. 1, 2, 3 and 4. These photo-micrographs, magnified 250 diameters, show the structure of the soft steel quenched respectively above Ar_3, between Ar_3 and Ar_2, between Ar_2 and Ar_1, and below Ar_1.

When quenched above Ar_3 the steel consists of 77 per cent. of martensite and 23 of ferrite. This martensite of course contains a very small proportion of carbon. It takes a light yellow color when etched with iodine or nitric acid, and is not very hard, being easily scratched by a needle. Between Ar_3 and Ar_2 the structure consists of 27 per cent. of martensite distributed as irregular grains throughout a matrix of structurally free ferrite. During the retardation at Ar_3, therefore, some 50 per cent. of the martensite has been replaced by ferrite. The martensite of this sample is much harder than that of the preceding one, and is colored quite dark by etching. It is very hard, standing prominently in relief, when the polishing is finished upon a yielding support, and cannot be marked by a needle. The grains of the ferrite, which are well shown in the drawing (Plate I., Fig. 2), are very faint in the photo-micrograph (Plate III., Fig. 2), owing to the difficulty of sharply focussing their fine junction-lines. Between Ar_2 and Ar_1 the amount of martensite has been reduced still farther, occupying now only 11 per cent. of the total mass. During the retardation Ar_2 some 15 per cent. of the ferrite, which above that point was enclosed in the martensite, has separated, bringing the amount of structurally free ferrite up to 89 per cent. The grains of the ferrite are shown imperfectly in the photo-micrograph, Plate III., Fig. 3. When quenched below Ar_1 we find that the structure is made up of 10 per cent. of pearlyte and 90 of

structurally free ferrite. During the critical point Ar_1, the 11 per cent. of martensite existing above that point disappears,

Fig. 4



MICRO-STRUCTURAL COMPOSITION OF STEEL QUENCHED ABOVE, BETWEEN, AND BELOW THE CRITICAL POINTS

⊙ Points actually determined

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yielding 10 per cent. of pearlyte and 1 additional per cent. of ferrite. This is also the structural composition of the same

steel when slowly cooled, and shows that quenching below the critical range changes neither the nature nor the relative amounts of the constituents. This is true for all steels. In the drawings (Plate I.) the pearlyte is shaded differently from the martensite. In the photo-micrograph (Plate III., Fig. 4) the pearlyte does not differ much from the appearance of martensite. We have seen how these two constituents, however, may be distinguished under the microscope. The grains of ferrite are well shown in the photograph (Plate III., Fig 4).

Steel No. 2—Carbon, 0.21 *Per Cent.*—In this steel the upper critical point Ar_3 is missing; and, when quenched before the retardation Ar_2 has set in, it is made up wholly of martensite (Plate I., Fig. 5). This martensite is, of course, not highly carburetted, and is only lightly colored by the etching. Between Ar_2 and Ar_1 the structure is made up of 31 per cent. of martensite in irregular grains, distributed throughout a matrix of structurally free ferrite (Plate I., Fig. 6). We infer from this structure that during the retardation Ar_2 some 70 per cent. of ferrite previously locked in the martensite has been set free. The grains of the ferrite are not shown in this drawing; they are in all respects similar to those outlined in Figs. 2, 3 and 4 of the same plate. Below Ar_1 we find that the mass is made up of 23 per cent. of pearlyte and 77 of ferrite (Plate I., Fig, 7), which shows that during the retardation Ar_1 the 31 per cent. of martensite existing above that point has been changed into 23 per cent. of pearlyte, setting free 8 per cent. more of ferrite.

Steel No. 3—Carbon, 0.34 *Per Cent.*—This steel, like the preceding one, has only two critical points, and, when quenched above Ar_2, contains only martensite (Plate I., Fig. 8), but a martensite somewhat more carburetted, therefore harder and more darkly colored. During the first retardation, Ar_2, 44 per cent. of ferrite is set free, reducing the percentage of martensite to 56 (Plate I., Fig. 9). Here the martensite no longer occurs in separate grains throughout the soft iron matrix, but forms an intricate continuous mass. Below Ar_1 the steel is made up of equal masses of pearlyte and ferrite (Plate I., Fig. 10). The character of the structure remains the same as if the martensite had merely been transformed *in situ* into pearlyte, yielding also a small amount (6 per cent.) more of free ferrite.

Steel No. 4—Carbon, 0.80 *Per Cent.*—Here we have reached

the saturation-point. The whole mass of the steel when slowly cooled or quenched below Ar_1 is made up of pearlyte (Plate I., Fig. 12). It has only one critical point, above which it is wholly composed of martensite (Plate I., Fig. 11). This martensite, which contains much carbon, is colored very dark and is extremely hard.

Steel No. 5—Carbon, 1.20 *Per Cent.*—When quenched above its single critical point, this steel contains, besides a matrix of martensite, about 6 per cent. of structurally free cementite, disseminated, worm-like, through the mass (Plate I., Fig. 13). Below Ar_1 the martensite has disappeared, having been replaced by pearlyte, which forms the meshes of a net-work of cementite, the latter constituent occupying about 8 per cent. of the total volume (Plate I., Fig. 14).

Steel No. 6—Carbon, 2.50 *Per Cent.*—The structure of this highly-carburetted steel above the critical point is composed of 80 per cent. of martensite and 20 of structurally free cementite (Plate I., Fig. 15). At Ar_1 the martensite is changed into 77 per cent. of pearlyte, bringing the amount of free cementite up to 23 per cent. (Plate I., Fig. 16). The structure of this steel in its unhardened state is also shown in Plate III., Fig. 5, under a lower magnification. In the photographs the cementite is white.

In order to show how closely the structural changes just described correspond with the retardation, Plate II., already published in the *Journal of the Iron and Steel Institute*,* has been reproduced here. In this investigation, 21 bars of steel No. 2 were heated to 970° C., then slowly cooled to a series of points near and within the critical range, from which they were suddenly cooled. In this manner some of the bars were quenched *during the retardations themselves*, thus revealing the progress of the corresponding structural changes. The microstructural composition of the various bars, together with their quenching-temperature, are shown in Table III.

These percentages are plotted in Fig. 5, which shows graphically the progress of the structural changes as the cooling-temperature falls past the critical points.

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* "Further Notes on the Hardening of Steel," Henry M. Howe and Albert Sauveur, *Jour. Iron and Steel Inst.*, 1896, No. 1, p. 170.

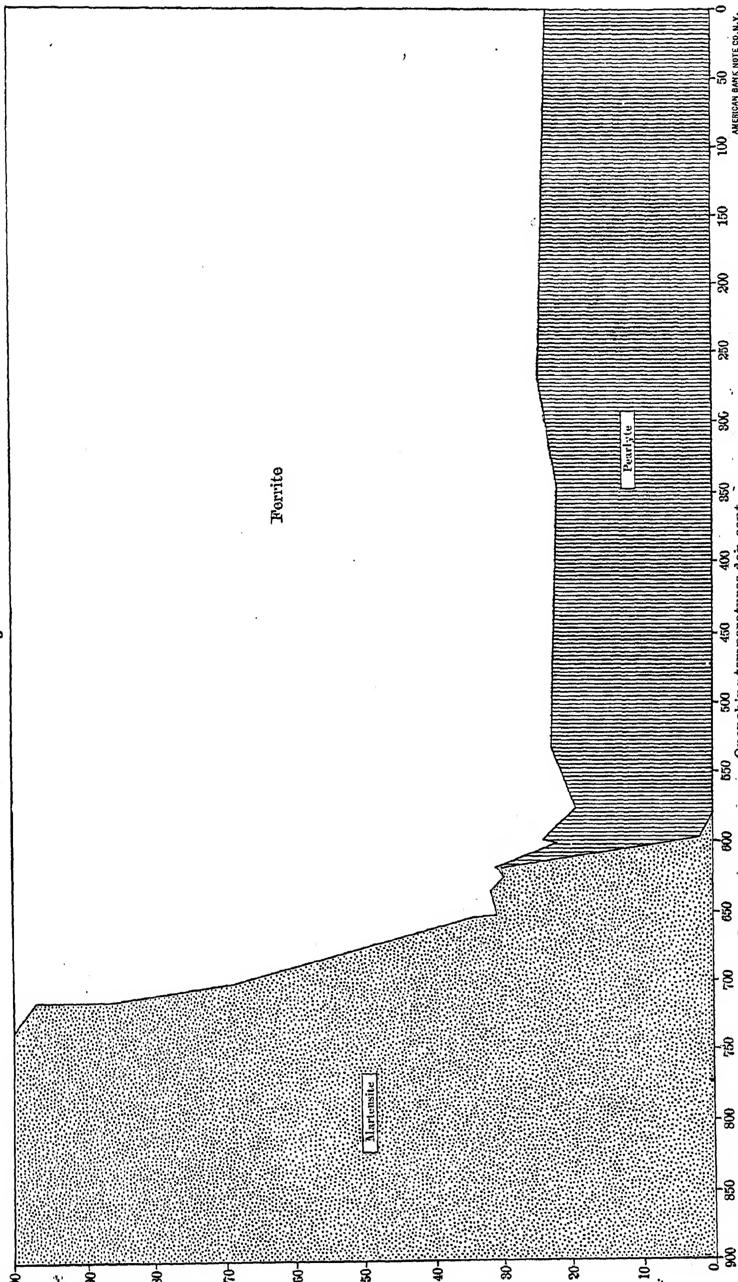
This graphical statement, as well as the figures of Table III. and the drawings of Plate II., show that structural changes occur *only during the retardations and that they begin and end with them.* The five bars quenched above Ar_2 all have the

TABLE III.—*Microstructural Composition of Steel Containing* 0.21 *Per Cent. of Carbon and Quenched Above, During and Below the Critical Range.*

Quenching-Temperature. Degrees C.	Positions of Quenching-Temperature.	Microstructural Compositions. Per Cent.		
		Martensite.	Ferrite.	Pearlyte.
880	Above Ar_2	100.	0.0	0.0
836	Above Ar_2	100.	0.0	0.0
797	Above Ar_2	100.	0.0	0.0
761	Above Ar_2	100.	0.0	0.0
733	Above Ar_2	100.	0.0	0.0
714	Beginning of Ar_2	97.20	2.80	0.0
713	Beginning of Ar_2	86.00	14.00	0.0
698	Middle of Ar_2	70.20	29.80	0.0
652	Between Ar_2 and Ar_1	35.20	64.80	0.0
650	Between Ar_2 and Ar_1	30.80	69.20	0.0
633	Between Ar_2 and Ar_1	32.00	68.00	0.0
626	Between Ar_2 and Ar_1	31.50	68.50	0.0
620	Beginning of Ar_1	30.00	68.40	1.60
600	End of Ar_1	4.00	78.50	17.50
559	End of Ar_1	2.00	75.80	22.20
575	Below Ar_1	0.0	78.90	21.10
532	Below Ar_1	0.0	76.80	23.20
512	Below Ar_1	0.0	77.00	23.00
340	Below Ar_1	0.0	77.40	22.60
263	Below Ar_1	0.0	75.20	24.80
200	Below Ar_1	0.0	76.40	23.60

same microstructure (Plate II., Fig. 1), made up entirely of martensite. As soon as the retardation Ar_2 begins, however, some ferrite is set free (Plate II., Fig. 2), increasing rapidly in amount as the cooling proceeds and the retardation becomes more marked. In the middle of Ar_2 we find about 30 per cent. of structurally free ferrite (Plate II., Fig. 4). Between Ar_2 and Ar_1 the metal contains some 70 per cent. of ferrite and 30 of martensite. The structure then remains unaltered until the second critical point is reached. The four samples quenched between Ar_2 and Ar_1 have a similar structure. This is well shown in Fig. 5. With the beginning of Ar_1 we find another structural change taking place, *i.e.*, the martensite begins to be replaced by ferrite; and the transformation, like the preceding one, con-

Fig. 5



MICRO-STRUCTURAL COMPOSITION OF QUENCHED STEEL CONTAINING 0.21% CARBON

tinues as long as the retardation lasts. At the very beginning of Ar_1 we find only 1.60 per cent. of pearlyte (Plate II., Fig. 2). At the end of that retardation the steel contains 22 per cent. of ferrite, only 2 per cent. of martensite remaining unchanged (Plate II., Figs. 7 and 8). Below Ar_1 there is no more martensite; the steel is made up of some 22 per cent. of pearlyte and 77 of ferrite. The seven bars quenched below Ar_1 show that no more change of structure takes place on farther cooling. The microstructural composition retained in steel by sudden cooling below the critical range does not differ from that of the slowly cooled metal.

Résumé.—Each critical point is accompanied by a structural change which begins and ends with it. In the ranges of temperature where there is no critical point, we find no change in the microstructural composition. Judging from the microscopical evidences, the changes which occur at the upper retardations, Ar_3 and Ar_2, are merely structural, and consist in the liberation of a certain amount of ferrite, which above those critical points was included in the martensite. The change which occurs at the lower point, Ar_1 or $Ar_{3,2,1}$ (in steels which have only one retardation) and which consists in the disappearance of martensite and the appearance of pearlyte, if it results, as is supposed, in the formation of Fe_3C, would indicate a chemical change.

Above Ar_3 (see Fig. 4) 0.12 parts carbon can unite with as much as 99.88 parts Fe (supposing, for simplicity, that the steel contains nothing but iron and carbon), or roughly, 1 part of carbon to 800 of iron. The martensite cannot absorb more ferrite; and if the steel contains less than 0.12 per cent. of carbon, which means a corresponding larger amount of iron, there is an excess of iron which remains unabsorbed as structurally free ferrite. Between Ar_3 and Ar_2 we find 0.25 carbon with 99.75 of iron, or in the proportion of 1 part of carbon to 400 parts of iron. In this range of temperature the martensite cannot assimilate more ferrite. The excess, if any, remains structurally free. Between Ar_2 and Ar_1 we find the amount of ferrite which can unite with carbon to form martensite is further lowered, being 0.50 carbon to 99.50 iron, or 1 part of carbon to about 200 of iron. If more ferrite is present it remains unabsorbed. Here we have what might be called the *saturation-point of carbon for iron;* and we see that on heating, each retardation

raises the saturation-point, causing a new structural arrangement. Fig. 4 illustrates also the fact that martensite never contains more than 0.90 per cent. of carbon, or 1 part of carbon to 110 parts of iron. If the steel is more highly carburetted, there is an excess of carbon, which remains unabsorbed as structurally free cementite. This may be called the *saturation-point of iron for carbon.* Martensite contains, therefore, all the way from 0.12 to 0.90 per cent. of carbon, according to the degree of carburization of the steel and the temperature at which it has been quenched.

The structural changes just studied may be explained as follows: If a piece of carbon-steel be heated beyond the critical point Ac_1, the iron and carbon which compose it combine, during this retardation, to form martensite, 1 part of carbon uniting with as much as 200 parts of iron (0.50 C; 99.50 Fe). If the steel contains more than 0.50 per cent. of carbon, therefore, the whole of the iron will be absorbed; and on further heating no other structural change takes place, nor do we find any other critical point. As martensite never contains more than 0.90 per cent. of carbon, we find, if more carbon is present, some structurally free cementite in the steel. If, on the other hand, the steel contains less than 0.50 per cent. of carbon, some iron still remains unabsorbed above Ac_1, as structurally free ferrite; and on further heating, we meet a second retardation, Ac_2, during which the saturation-point of carbon for iron is raised to 1 part of carbon to 400 parts of iron (0.25 C; 99.75 Fe). During this retardation a certain additional amount of ferrite is absorbed by the martensite, and a new condition of structural equilibrium is established. If the steel contains more than 0.25 per cent. of carbon, the whole of the free ferrite will consequently disappear during Ac_2, and further heating will reveal no more critical points or structural changes. Between 0.25 and 0.50 per cent. of carbon the steels have only two critical points. Below 0.25 per cent. of carbon some ferrite still remains unabsorbed above Ac_2. Mild steels have a third retardation, Ac_3, which raises the saturation-point of carbon to 1 part carbon to 800 parts of iron (0.12 C; 99.88 Fe), so that the whole or a portion of the remaining ferrite is again assimilated by the martensite. If more than 0.12 per cent. of carbon is present, the whole of the ferrite disappears; if less than 0.12 is present,

some ferrite will remain unabsorbed on further heating. The reverse phenomena occur during slow cooling at Ar_3, Ar_2, and Ar_1. The ferrite which was absorbed at Ac_3 and Ac_2 is again set free at Ar_3 and Ar_2, while at Ar_1 the martensite is again replaced by pearlyte.

II.—The Current Theories of Hardening.

Assisted by the microscopical evidences presented in the first part of this paper, the writer proposes to review critically the various theories which have been advanced to explain the hardening of steel.

Mr. Howe very clearly sums up the two main theories in the following words:*

> "Both these theories are of the *status quo* kind, *i.e.*, each holds that the sudden cooling acts by preserving a spec'al condition which exists at a red heat, but is unstable at lower temperatures, and does this by denying the time which is required for the change from that condition to the normal condition stable at the common temperature, and such as we find in annealed steel. At and above a certain critical range, ... the a, of Osmond, ... carbon passes spontaneously from the normal condition called "cement-carbon," to a special condition called "hardening-carbon," which below that range is unstable, tending to pass back to the cement-state, and actually so passing back during slow cooling. It may, however, be preserved by sudden cooling, and hence is found in hardened steel. . . . . While the carbon-theory regards this known preservation of carbon in the hardening state as the direct cause of hardening, the allotropic theory holds that its action is indirect; that above the critical range the iron passes spontaneously to a special strong, hard, brittle, allotropic state called β-iron, which, like hardening carbon, is unstable below that range, but which, still like hardening carbon, passes back relatively slowly from the β to the α state, so that the β state may be preserved by sudden cooling."

Mr. Howe himself has tried to reconcile these two opposite theories by attributing the hardening of steel to the existence at a high temperature, and the retention by sudden cooling, of a carbide of an allotropic form of iron; and he calls his theory the "carbo-allotropic" theory.

Prof. Arnold, in a remarkable paper† recently read before the British Institution of Civil Engineers, attributes the hardening of steel, not to the carbon as such, but to the existence

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* "The Hardening of Steel," H. M. Howe, *Jour. Iron and Steel Inst.*, 1895, No. 2, p. 258.

† "The Influence of Carbon on Iron," J. O. Arnold. *Proc. Inst. Civil Eng.*, vol. cxxiii., London, 1896, p. 127.

Table IV.

	Carbon Theory.	Allotropic Theory.	Carbo-Allotropic Theory.	Prof. Arnold's Sub-Carbide Theory.		Microstructural Evidence.
Conditions of carbon and iron above Ar_3.	Iron and hardening-carbon.	Carbon diffused in γ iron.		Carbon and iron combined as $Fe_{24}C$.	?	Very mild steel, made up of martensite and ferrite.
Cause of the evolution of heat at Ar_3.	?	Passage of the iron from the γ to the β state.		Dissociation of $Fe_{24}C$.	Formation of $Fe_{24}C$.	Separation of a certain amount of ferrite previously included in martensite.
Conditions of carbon and iron between Ar_3 and Ar_2.	Iron and hardening-carbon.	Carbon diffused in β iron.		?	Carbon and iron combined as $Fe_{24}C$.	Soft and medium hard steel made up of martensite and ferrite.
Cause of the evolution of heat at Ar_2.	?	Passage of the iron from the β to the α state. Steel becomes magnetic.		Passage of iron from a plastic to a crystalline condition.		Separation from the martensite of an additional amount of ferrite.
Conditions of carbon and iron between Ar_2 and Ar_1.	Iron and hardening-carbon.	Carbon diffused in α iron.		?	Carbon and iron combined as $Fe_{24}C$.	Soft steel: martensite and ferrite. Medium hard steel: martensite. Hard steel: martensite and cementite.
Cause of the evolution of heat at Ar_1.	Passage of carbon from hardening- to cement-state.	Carbon combining with α iron to form Fe_3C.		Carbon combined with iron to form Fe_3C.	Dissociation of $Fe_{24}C$ and formation of Fe_3C.	Disappearance of martensite and appearance of pearlyte.
Conditions of carbon and iron below Ar_1.	Iron and cement-carbon combined as Fe_3C.	Iron and cement-carbon combined as Fe_3C.		Iron and carbon combined as Fe_3C.		Soft steel: pearlyte and ferrite. Hard steel: pearlyte and cementite.
Cause of hardening.	Retention by sudden cooling of the carbon in its hardening-state.	Retention by sudden cooling of a hard allotropic state of iron, which retention is helped by the presence of C.	Retention by sudden cooling of a hard carbide of γ or β iron.	Retention by sudden cooling of a hard sub-carbide of iron, $Fe_{24}C$.		Retention of the martensite by sudden cooling.

above the critical range of a very attenuated and very hard carbide of iron. This will be here called the *subcarbide theory.*

Each theory accounts in its own way for the critical points, and attributes to the carbon and iron above, between and below the retardations such conditions as are in accordance with its claims. These facts, as well as the microscopical evidences, are summarized in Table IV.

As we have already stated, martensite is the constituent which confers hardness upon suddenly cooled steel. If we knew its composition, the mist which now surrounds the phenomenon of hardening would be cleared away. Unfortunately, the details of its structure are so minute and so difficult to resolve that, even when most highly magnified, it gives little indication of its chemical composition or even structural character. From its mode of occurrence, however, as already described, we are able to infer certain of its characteristics which have an important bearing upon the theories just outlined.

As it has been so plainly shown, martensite contains all the way from 0.12 to 0.90 per cent. of carbon, according to the grade of the steel and the temperature from which the same has been quenched. We immediately conclude that martensite cannot be a chemical union of carbon and iron, as Prof. Arnold would have it, unless it were argued that it is a carbide of iron whose formula varies with the amount of carbon contained in the steel, and also at each critical point, from $Fe_{178}C$ to $Fe_{24}C$, which would be absurd. The writer does not see how it is possible to avoid the conclusion that the carbon of the martensite is diffused or dissolved in the iron. At each critical point, on heating, the power of diffusion of the carbon (or, as it has been called, its saturation-point for iron) is increased, so that it diffuses through a larger mass of ferrite. The question whether the carbon is diffused as such or whether it first combines chemically at a high temperature with a definite proportion of iron, forming a true carbide, which then diffuses through the iron, remains, however, undecided.

The Carbon-Theory.—This theory, which for so many years satisfied the steel metallurgists, is somewhat crude and incomplete, attributing in a vague manner the hardening of steel to the existence at a high temperature of carbon in a hard state, and to its retention by sudden cooling. The theory owes its

existence chiefly to the fact that when hardened steels are dissolved in cold dilute acids, nearly all the carbon escapes in the form of hydrocarbons, while unhardened steels, when similarly treated, yield nearly the whole amount of their carbon as a residue, the composition of which corresponds to the formula Fe_3C. From this behavior of the carbon, it was reasonably inferred that in hardened steel the carbon existed in a different form from that of the unhardened metal. What this hardening-state of the carbon really was, has never been satisfactorily shown. If the theory implies that it is to the carbon as such that quenched steel owes its hardness, is it not difficult to understand how, say 0.50 per cent. of carbon, be it as hard as it may, distributed throughout 99.50 per cent. of a matrix of soft iron, can produce the glass-like hardness of a steel of this composition when suddenly cooled above the critical range?

Prof. Arnold's effort, and its resulting sub-carbide theory, cannot be regarded as a mere modification of the carbon-theory, and will be considered separately.

Allotropic Theory.—In this theory, so ably worked out by Mr. Osmond and supported by Prof. Roberts-Austen, the hardening of steel is attributed to the retention in the suddenly-cooled metal of a hard allotropic state of the iron. The allotropists had, however, to concede at the outset the all-important part which carbon takes in the hardening of steel. As Mr. Hadfield has remarked, take away the carbon and the hardness disappears. No carbonless alloy of iron has ever been produced which exhibited anything like the extreme hardness of quenched high-carbon steel, or which was susceptible of being materially hardened by sudden cooling. To reconcile this with their theory, the allotropists argue that the carbon assists the retention of iron in its hard allotropic state, in this way accounting for the fact that the degree of hardness produced by sudden cooling is roughly proportional to the amount of carbon. While such reasoning cannot well be refuted, it lacks cogency, and in some way predisposes us against the allotropic theory.

The allotropists say, moreover, that below Ar_2 (see Table IV.) the iron exists in the α state, passing to the hard β variety on heating at Ac_2. How then do they explain the hardness of steel quenched between Ar_2 and Ar_1, seeing that the metal in this range of temperature contains only soft α iron? The hard-

ening-power is not lost, as they would have it, as soon as the steel is cooled below Ar_2. Mr. Howe has shown that the loss of hardening-power, instead of coinciding with the upper retardation, lags even behind the lower point Ar_1. There is, indeed, very little doubt that it is during the retardation at Ac_1 that steel acquires most of its hardening-power.

According to Mr. Osmond, martensite represents "the crystallic organization (under the influence of carbon) of one of the allotropic varieties of iron." Above Ar_3 we have what might be called γ martensite; between Ar_3 and Ar_2, β martensite; below Ar_2, α martensite. This last variety, since it contains only α iron, should be very soft, but, as a matter of fact, it is very hard, harder than the β or γ martensite. The difference of hardness between the martensite of soft steel and that of more highly carburetted metal is not due, according to the allotropists, to the greater amount of carbon contained in the latter variety, but to the greater proportion of β or γ iron, which a larger amount of carbon retains in the martensite when suddenly cooled. All martensite contains some α iron, but in proportion decreasing with its carbon-content.

From the fact that the structurally free ferrite found in mild steel when quenched is very soft, we infer that it is α iron, and therefore that the hard variety β or γ never exists structurally free in the cold metal, being present only in the martensite.

When heated to a high temperature (above the critical range), steel loses its power of being attracted by a magnet, from which it is argued that while α iron is magnetic, β and γ irons are not magnetic. Suddenly-cooled steels, however, are magnetic. This, the allotropists contend, is due to the fact that a portion of the non-magnetic β or γ iron passes back to the magnetic α state, however rapid the cooling. But then, since the more carbon present in the steel, the less α iron should be found in the quenched metal, high-carbon steels should be considerably less magnetic than mild steels, while, as a matter of fact, they are more magnetic. Again, from the fact that manganese steel (containing some 12 per cent. of manganese), when slowly cooled, does not exhibit any critical point, and is both hard and non-magnetic, the allotropists conclude that manganese, when present in considerable quantity, prevents the

passage of iron from the hard, allotropic variety to the soft α state. Since manganese steel is non-magnetic, it cannot contain any α iron, and should therefore, according to their theory, be harder than quenched carbon steel; but, on the contrary, it never approaches the extreme hardness of high-carbon steel suddenly cooled.

It is claimed that the very existence of the critical points is a proof of allotropy, but the allotropists themselves admit that the lower retardation at Ar_1 is due to a chemical change. Here, then, at least, is an evolution of heat which is not of allotropic origin. The existence of the critical points, therefore, does not prove allotropy, however suggestive of such a change it may be. It seems to the writer that the two strongest arguments which have been put forward in favor of the allotropic theory have been, first, the appearance and disappearance of magnetism, which, according to Mr. Osmond and Prof. Arnold, correspond to the reversible changes at Ar_2 and Ac_2;* and, second, the fact that nearly carbonless iron presents marked retardations at Ar_3 and Ar_2. But the opponents of allotropy have claimed that absolutely pure iron will not present these two critical points, and that whenever they have been found there was at least a few hundredths of 1 per cent. of carbon present as well as a relatively considerable amount of other impurities (especially sulphur). Prof. Roberts-Austen, however, detected these two retardations in absolutely carbonless iron.†

Prof. Arnold's Sub-Carbide Theory.—This theory, presented with a profusion of evidence,‡ microscopical, chemical, mechanical and physical, all skilfully brought to the desired focus, has been implicitly accepted by many metallurgists, who concluded that the problem of the hardening of steel had at last been solved. In the introduction of his paper, Prof. Arnold expresses himself as follows: "If it can be proved that at high temperatures the carbon still remains in combination with

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* Mr. H. Tomlinson, an authority on such matters, finds that the magnetic change occurs at about 850° C. in nearly carbonless iron, at about 750° C. in mild steel, and 650° C. in hard steel. This would make the appearance and disappearance of magnetism correspond with the upper retardation in all steels, *i.e.*, with Ar_3 or $Ar_{3,\,2}$ or $Ar_{3,\,2,\,1}$. (See Fig. 1.)

† *Proc. Inst. of Mech. Eng.*, London, 1895, p. 244.

‡ "The Influence of Carbon on Iron," J. O. Arnold, *Proc. Inst. Civil Eng.*, vol. cxxiii., Part I., 1896, p. 127.

the iron, the foundation of the β iron theory will be destroyed, and its superstructure must naturally collapse." The allotropists would probably take exception to this conclusion; but let us see how Prof. Arnold proceeds to prove the existence at a high temperature of an iron-carbide of the very improbable formula $Fe_{24}C$. He argues that since a steel containing about 0.90 per cent. of carbon, when slowly cooled, is entirely made up of pearlyte, and when quenched above the critical range contains nothing but martensite, this martensite must be a very attenuated carbide containing 0.90 per cent. of carbon and 99.10 of iron, and therefore corresponding to the formula $Fe_{24}C$ (which requires 0.884 per cent. C). This Prof. Arnold says is the only possible way we can account for the presence of pearlyte in slowly cooled steel. "Unless," he says, "a sub-carbide existed, which, before the critical point Ar_1 was reached, gathered itself into masses distinct from the iron, there does not appear to be any reasonable explanation why the Fe_3C was not evenly diffused through the iron of normal unsaturated steel."

The writer does not see on what grounds Prof. Arnold, who concedes to his sub-carbide the right to segregate into distinct masses above Ar_1, denies the same privilege to the diffused carbon of the allotropists. It is furthermore useless to point out again the fact already conclusively established that martensite cannot be a true chemical union of carbon and iron. Will Prof. Arnold contend that the carbide $Fe_{24}C$, which forms the only constituent of hardened steel containing about 0.90 per cent. of carbon, is, in softer steel, diffused in a certain amount of iron, the softer the steel the greater being the degree of diffusion? It is the only possible way to reconcile his theory with the microscopical evidences. To further establish the existence of his sub-carbide, Prof. Arnold reasons about as follows: At 0.90 per cent. of carbon, which he calls the saturation-point of the steel, the quenched metal is entirely composed of sub-carbide; below 0.90 it contains some iron; above this it contains some free Fe_3C; therefore, if it can be shown that in a series of steels containing increasing amounts of carbon, when 0.89 per cent. of carbon is reached, a critical point occurs in the mechanical, physical and chemical behavior, we will have additional proof of the existence of the sub-carbide. The

writer cannot see what bearing this has upon the existence of a sub-carbide. Let us see if, upon closer scrutiny, Prof. Arnold's conclusions become more convincing. He says: "If the author's (Prof. Arnold's) theory be true, it demands the somewhat startling theoretical condition that the maximum heat at Ar_1 should be evolved from iron containing 0.884 per cent. of carbon;" and this he finds to be actually the case. Far from being startling, however, the reason for this is quite evident, and is a natural sequence of the structure of the steel. It has nothing whatever to do with Prof. Arnold's theory. If we refer to his drawings or to Plate I. of the present paper (here the saturation-point occurs at 0.80 C), it is seen that when the steel is saturated, *the whole mass* changes its structure and possibly its chemical composition during the retardation at Ar_1, while with a softer or harder steel only a portion of the total volume is affected in this way. Is it not evident, therefore, that the saturated steel should evolve more heat than the lower or higher members of Prof. Arnold's series? At about 0.90 per cent. C (0.80 in commercial steel) we have a critical structural point; and it is quite natural to expect correlative critical points in the physical and mechanical behavior of the metal. This fact, upon which Prof. Arnold lays so much stress, does not point, even remotely, toward the existence of his sub-carbide.

Prof. Arnold uses the notation Ar_1, Ar_2 and Ar_3 to designate the retardations, whether they occur in heating or in cooling; and on this account it is not clear whether he claims that his sub-carbide forms during heating at Ac_3 or during cooling at Ar_3. Prof. Arnold says: "Molten masses of hard steel consist of a mixture of the normal carbide Fe_3C with the sub-carbide." From this we would naturally infer that since the sub-carbide exists in the molten metal, it must be formed at Ac_3 (on heating), dissociating at Ar_3 (during cooling). On another page, however, he expresses himself as follows: "The point Ar_3 marks the formation of a sub-carbide, while the point Ar_1 is due to the combination of the elements to form the normal carbide Fe_3C." Here he certainly means that the sub-carbide forms during cooling. In presence of this contradiction, let us examine both hypotheses. If the sub-carbide is formed at Ac_3 (or $Ac_{3,2,1}$) on heating, what then is the condition of the carbon and iron between Ar_1 and Ar_3, seeing that Fe_3C disso-

ciates at Ac_1? and how is the hardening power of the steel in this range accounted for since we have here no sub-carbide?

If the sub-carbide forms at Ar_3 (or $Ar_{3,2,1}$) during cooling, the conditions are still more startling, for then steels quenched above Ar_3 or $Ar_{3,2,1}$ should not be hardened, since they contain no sub-carbide. The writer would suggest, as the only possible way out of this dilemma, that the sub-carbide forms on heating at Ac_1 and diffuses throughout the iron at Ac_2, and still farther at Ac_3. This would be supported by the structural evidences, but it is altogether opposed to Prof. Arnold's view, for he has always contended that the point Ar_3 or Ac_3 marks the formation of the sub-carbide.

Prof. Arnold says that the point Ar_2 marks the passage of iron from a "plastic to a crystalline condition." Ar_2 he calls the crystallization-point of iron. From this we necessarily infer that above Ar_2 the structure of iron is amorphous; for Prof. Arnold's assertion implies that at this point the iron passes from a non-crystalline to a crystalline structure, and a structure must be either crystalline or amorphous. Mr. Osmond's and the writer's microscopical investigations have conclusively shown that the size of the crystals or grains increases directly with the temperature from which the metal cools, and inversely with the rate of cooling; whereas, if Prof. Arnold were right, since the structure remains amorphous until Ar_2 is reached, it would make little difference how far above this point the steel were heated; provided it were heated above it, it would always have the same sized grains. Again, steel quenched above Ar_2 has a decided crystalline structure (Plate I., Fig. 2), while, according to Prof. Arnold it should be amorphous, unless it were argued that it crystallizes at Ar_2 during sudden cooling, but we know that quenched steel does not exhibit a retardation at Ar_2 or any other critical points. Indeed, is it not quite certain that crystallization sets in as soon as the metal is allowed to cool undisturbed, no matter how high the temperature? Can we conceive of a molten mass solidifying and remaining amorphous during cooling for a considerable time until a certain critical point is reached, when it crystallizes quite suddenly? Is not solidification in some way a synonym of crystallization, and with one or two exceptions (like glass and opal) is there in nature any substance which solidifies without crystallizing?

Prof. Arnold's contention is based upon the following experiment: A piece of iron was rolled to a low temperature, and upon being examined microscopically, it was found that the crystals were distorted and elongated in the direction of the rolling. Upon reheating this metal to 650° C. no change of structure was detected, but when heated to 750° C. the crystals were found to be again normal, all trace of distortion having disappeared. From this he infers that iron crystallizes during the retardation at Ar_2. What this experiment really told is, that the metal becomes amorphous at Ac_2 *during heating*, recrystallizing on subsequent cooling; but it does not mean that it remains amorphous *during cooling* until the point Ar_2 is reached. Now, while, as indicated above, the contention that steel remains amorphous *during undisturbed cooling* is quite untenable, we have conclusive evidence that the metal becomes amorphous during the critical range, and strong indications that it *remains so during heating*. For if a piece of steel slowly cooled from a high temperature, and therefore presenting large crystals or grains, be reheated just above the critical range and again allowed to cool we find the structure made up of very small crystals. The steel has evidently become amorphous during the retardation, and we cannot conceive of its crystallizing again during further heating. The fact that steel cannot remain amorphous during cooling, but does remain so during heating, may at first appear somewhat startling, but is it opposed to the laws which govern the formation and growth of crystals? Does not a falling temperature promote the formation of crystals; indeed, is it not a necessary condition to their formation? Does not a rising temperature, on the contrary, oppose it? During undisturbed cooling cohesive attraction acts powerfully, causing the crystals to grow. During heating, on the contrary, cohesive attraction is opposed more and more vigorously by the rising temperature, which finally destroys the crystalline arrangement and causes the structure to become amorphous. In other words, can crystallization take place during heating?

Prof. Arnold says that if steel crystallized at a higher temperature than 750° C. (Ar_2), say at 900° C., it would imply that all irons quenched at a temperature lower than 900° C. would then have their crystals elongated in the direction of the roll-

ing. As a matter of fact, however, *undisturbed* cooling is necessary for the crystallization to proceed. If the metal was forged or rolled, it possesses at the end of the operation an amorphous structure (provided the forging has been sufficiently vigorous). If then the temperature is so low that crystallization cannot set in, the structure remains amorphous. Dr. Sorby was the first to remark that the grains of iron considered as forming each a crystalline individual are not drawn out in rolling, and are therefore constituted after the metal has passed the rolls. The reason that Prof. Arnold finds elongated crystals in this sample must be attributed to an insufficiently vigorous kneading or to the fact that the steel had not been heated to a sufficiently high temperature previous to rolling, *i.e.*, that it had not been heated past the critical range, so that the structure was crystalline at the beginning of the rolling, and the crystals had too much rigidity to be effaced by the pressure of the rolls, being susceptible only of undergoing a distortion in the direction of the rolling.

Prof. Arnold, as already stated, does not use the word "amorphous," but employs the word "plastic." Still he speaks of Ar_2 as the crystallization-point of iron. The crystallization-point of a substance is that point at which it passes from an amorphous to a crystalline condition; it has no other meaning. It cannot be that Prof. Arnold means that at Ar_2 the iron passes from a plastic to a non-plastic or rigid condition. Plasticity is the power possessed by certain substances of being molded. This change could hardly cause an evolution of heat. On heating the iron becomes plastic very gradually, becoming more and more so as the temperature rises. Plasticity is a relative quality. There is no narrow range of temperature of which it can be said that above it the iron is plastic, while below it is non-plastic.

There are many phenomena in the physics of steel which cannot be accounted for by any of the theories advanced, and in presence of the confusion which so much speculation is likely to throw on this subject, it is of great importance to keep carefully in mind the facts which have been proved beyond reasonable doubts. They may be classified as follows:

1. Carbon-steels during cooling or heating present one or more critical points or retardation. The retardations on heating are the reversals of the corresponding retardations on cool-

ing, *i.e.*, one cannot take place unless the other has been induced since the former last took place. They are therefore only the opposite phases of the same phenomenon. Very mild steels have three such retardations; mild and medium hard steels have two retardations; hard steel only one. Ac_3 and Ac_1 occur at a temperature somewhat higher than Ar_3 and Ar_1, while Ac_2 and Ar_2 occur nearly at the same temperature.

2. When a carbon-steel is quenched from a temperature above the critical range it is hardened in a degree roughly proportional to the amount of carbon. When quenched below the critical range it is not sensibly hardened.

3. Carbon-steel heated above the critical range is non-magnetic; below the critical range it is magnetic.

4. When hardened steel is dissolved in cold dilute acids the greatest portion of its carbon escapes as hydro-carbons. When unhardened steel is similarly treated nearly all of the carbon is found in a residue corresponding to the formula Fe_3C.

5. Each critical point is accompanied by a marked change in the microstructural composition of the metal, and no such changes occur except at the critical points.* During the two upper retardations a certain amount of iron, previously combined with the carbon, and existing as a distinct constituent, martensite, is set free, thus increasing the degree of carburization of the remaining martensite and reducing it in quantity. During the second retardation a similar change occurs resu'ting in a further reduction of the volume occupied by the martensite and a further increase of its percentage of carbon. During heating opposite changes occur at Ar_3 and Ar_2, the amount of ferrite set free at the corresponding point during cooling being here reabsorbed by the martensite. During the retardation at Ar_1 the martensite is again slightly reduced in volume, and its structure entirely changed, being transformed into the constituent which has been described as pearlyte, and which is made up of a structural mixture of iron and the carbide Fe_3C. During heating at Ac_1 the reverse change takes place. The pearlyte changes back to martensite, absorbing besides a small additional amount of iron.

* The size and shape of the grains of steel vary during slow cooling with the initial temperature and the rate of cooling, but these structural changes are not changes of microstructural composition. They affect neither the nature of the constituents nor their relative proportions.

6. Manganese-steel containing about 12 per cent. of manganese does not show any marked retardation during slow cooling. It is hard and non-magnetic.

Forgetting for the present the theories just reviewed, and with only the direct structural evidences before us, let us see if we can account for the various phenomena. We have seen that not only is every critical point accompanied by a change of microstructural composition, but also that each such structural change is accompanied by a retardation. Is it unreasonable, therefore, to infer that the absorption or evolution of heat which occurs at the critical points is due to the structural changes themselves? The saturation-point of carbon for iron is lowered at the two upper critical points on cooling, resulting in the separation of a certain amount of iron. Cannot this change produce an evolution of heat? Is it not the passage of a substance from a less to a more stable condition? If it be true that the carbide Fe_3C does not exist above Ar_1, the lowest retardation is both of structural and chemical origin, and the evolution of heat is here easily explained. But can the hardening of steel be accounted for without calling in allotropy or the existence at a high temperature of a hard state of the carbon or of a hard carbide? We know that the hard constituent of steel, martensite, is made up of all the carbon present in the steel (except in very hard steel where there is an excess of carbon present as cementite) with a portion or the whole of the iron. We know, moreover, that whatever the condition of the carbon (whether it exists as such or as a carbide) it is diffused in the iron of the martensite, and capable of diffusing throughout a larger mass during the retardation on heating. Now the carbonists claim that it is carbon in a hardening-state which is dissolved through the iron, while the allotropists contend that the carbon is diffused through a mass of a hard allotropic state of the iron. The writer will venture to suggest that it might be the normal carbide Fe_3C which is thus diffused through the iron, though the different behavior of the carbon of hardened and unhardened steel, when dissolved in cold dilute acids, seems at first to render such supposition impossible. If the carbon is still combined with iron at a high temperature, as Fe_3C, this carbide must be diffused through the iron, and is it not possible that the escape of the carbon, on

Plate III.

Fig. 1.

Fig. 2.

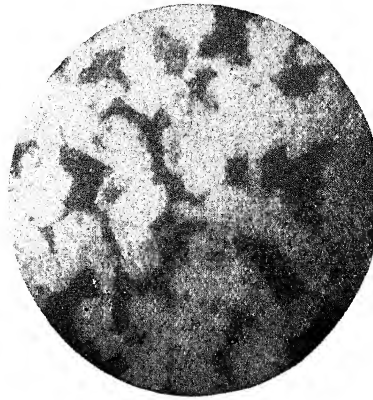


Fig. 3.

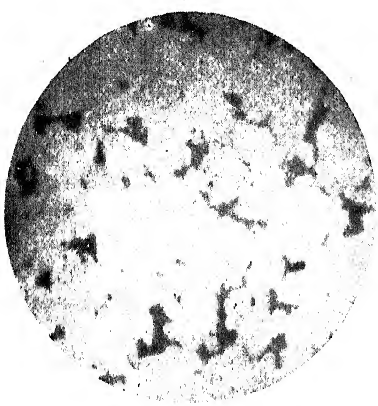


Fig. 4.

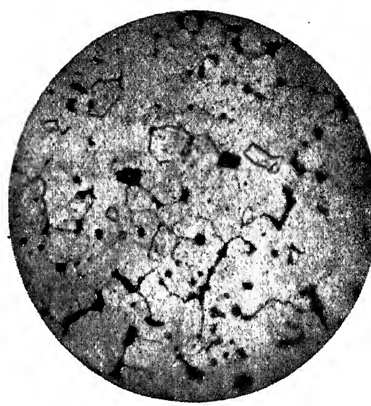


Fig. 5.

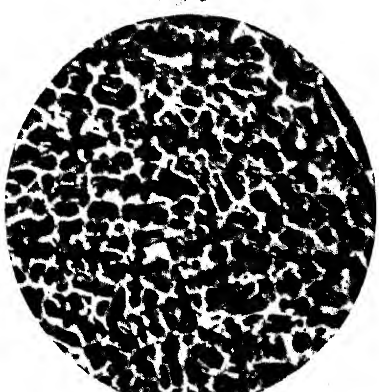
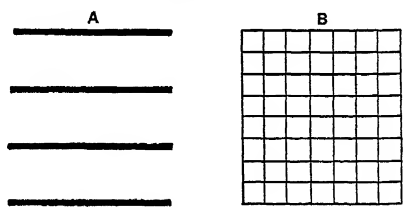


Fig. 6.

dissolving hardened steel, is due to the extreme degree of diffusion of the carbide Fe_3C? In the residue from the solution of unhardened steel the carbide Fe_3C is found in comparatively large plates and grains, such as can be detected through the microscope in the pearlyte (lamellar and granular). But in the martensite of hardened steel cannot this Fe_3C exist in such an extremely minute state of division as to account for the volatilization of its carbon? The writer asks whether this is opposed to chemical laws, and hopes that the chemists will give their opinions upon this point.

The hardness of martensite can very well be attributed, then, to the diffusion through its mass of the carbide Fe_3C, which we

FIG. 6.



know exists in steel below the critical range, and which we *know* to be extremely hard. It will no doubt be argued that if the hardness of martensite is produced by the presence of a certain amount of Fe_3C, or cementite, then the pearlyte of slowly cooled steel should be equally hard. Let us consider, for instance, the case of a steel which below the critical range is composed entirely of pearlyte, while above this it contains only martensite. Here the martensite and pearlyte would have the same chemical composition; they would have the same proportion of iron and Fe_3C. In pearlyte, however, the iron and the carbide each segregates in comparatively large masses, while in martensite the hard Fe_3C is uniformly diffused through the whole mass. May not this account for the difference in their mineralogical hardness? To make this matter clear, the structure of pearlyte, which contains 1 part of Fe_3C to 7 parts of iron, has been sketched in A, Fig. 6, while the structure of martensite can be imagined as appearing somewhat like B.

A and B show the same proportion of intensely hard Fe_3C

and of soft iron, but on account of the different structural arrangement and distribution of these two constituents, will not the hardness of martensite, or, rather, its power of resisting abrasion, be much greater than that of pearlyte? The relatively large soft areas of the pearlyte will be easily scratched or worn away by friction, while the soft meshes of the martensite are on all sides protected by the hard carbide, thus presenting great resistance to abrasion. The soft meshes may actually be so minute that even the point of a needle will be prevented by the surrounding cementite from marking them.

The writer does not wish to advance here any new theory. Indeed, if he knew that his remarks would be so interpreted he would refrain from giving them shape. He only feels that they may contain some element of truth which may help in solving the problem discussed in this paper, and if his arguments and deductions are not sound he hopes they will be promptly refuted.

The Enterprise Mine, Rico, Colorado.

BY T. A. RICKARD, STATE GEOLOGIST, DENVER, COLO.

(Colorado Meeting, September, 1896.)

I.—Historical.

Rico, in the southwestern corner of Colorado, is one of the productive mining centers of the San Juan region, so-called because its waters drain into the river of that name, which is tributary to the Colorado. The San Juan region includes the counties of Ouray, Hinsdale, San Miguel, Dolores, San Juan and Montezuma. It is traversed by a network of picturesque mountain ranges on whose lofty summits there rests perpetual snow. The region is peculiarly rugged, and, in the early days of its development, tested to the full the hardihood of the adventurers who first explored its cañons in search for gold and silver.

A prospecting party, guided by Jim Baker, scout and trapper, penetrated in 1861 this part of the territory of Kansas. At that time the country was in the possession of the Ute Indians. In October, 1873, by the Brunot treaty, they ceded to the United States Government the richest mineral-bearing portion of their

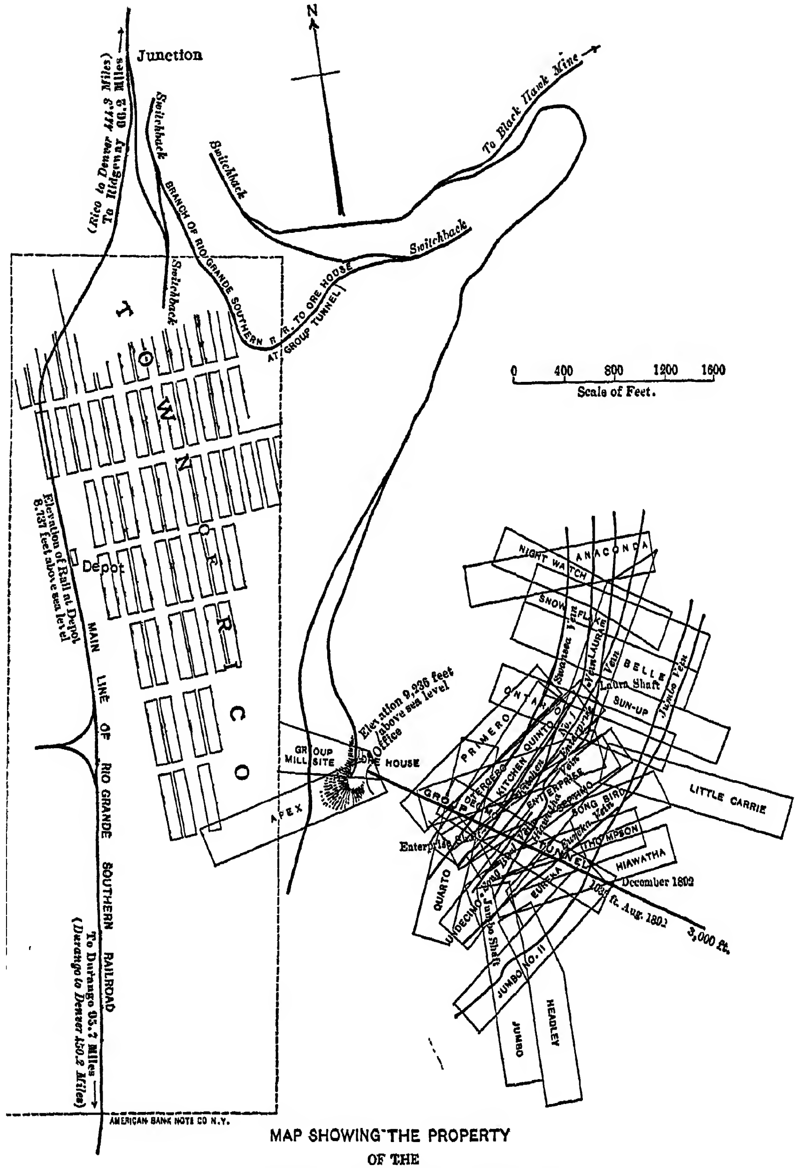
domain. But in the interval much prospecting had already been done, in defiance of difficulties among which snowslides and redskins were the most noteworthy. The mountains bordering the Animas and its tributaries were first explored by the pioneers, but the gathering wave of immigration soon swept further westward, and in 1864 a guide named Robert Darling brought a party of United States army officers and Mexicans from Santa Fé to the croppings of certain lodes which he had found on the Dolores* river. This party erected an adobe furnace and spent an entire summer in an abortive attempt to smelt the ores, the outcrop of which can still be seen at the north end of the main street of the town of Rico, upon claims now owned by the Atlantic Cable Company. In the autumn they returned to Santa Fé, and the valley was given up to the trappers and hunters, who found beaver along the stream and bear and deer on the hillsides.

In 1869 another expedition arrived. It consisted of John Eckels, William Hill, Pony Whitmore and two others, all of whom had made their way from the Moreno mines, a district near Elizabethtown, in what is now New Mexico. They discovered several large lodes near the site of the present settlement of Dolores. In the following year, Gus Begole came across the range from Silverton, and brought an assay-outfit with him. He and his partner, Eckels, discovered and located the Nigger Baby (now Yellow Jacket) and Dolores (now Aztec) mines. They sank several shafts and ran several drifts, but the ore proved too low in value to meet the costs of treatment and transportation, and they abandoned their claims. Others, who came from time to time, had a like experience.

In 1878, John Glasgow, Charles Hummiston and Sandy Campbell found their way northward from La Plata City. They spent the summer in active work, and located the Atlantic Cable, Grand View, Phœnix, Yellow Jacket and other claims. During the succeeding winter and in the early spring of 1879, the news went out that "carbonates" had been found

* If the Spaniard devastated the countries he conquered, he at least left a poetic nomenclature in his wake. The river Animas was called Rio de las Animas perdidas—"the river of lost souls," and the gloomy magnificence of its tumultuous way renders the name appropriate. Dolores, Durango, San Miguel, San Juan, Ignacio, Dulce, Juniata, etc., compare well with Cripple Creek, Leadville, Central City, Corkscrew, Coke Ovens, etc.

Fig. A.



MAP SHOWING THE PROPERTY

OF THE

ENTERPRISE MINING COMPANY

AT RICO, COLORADO.

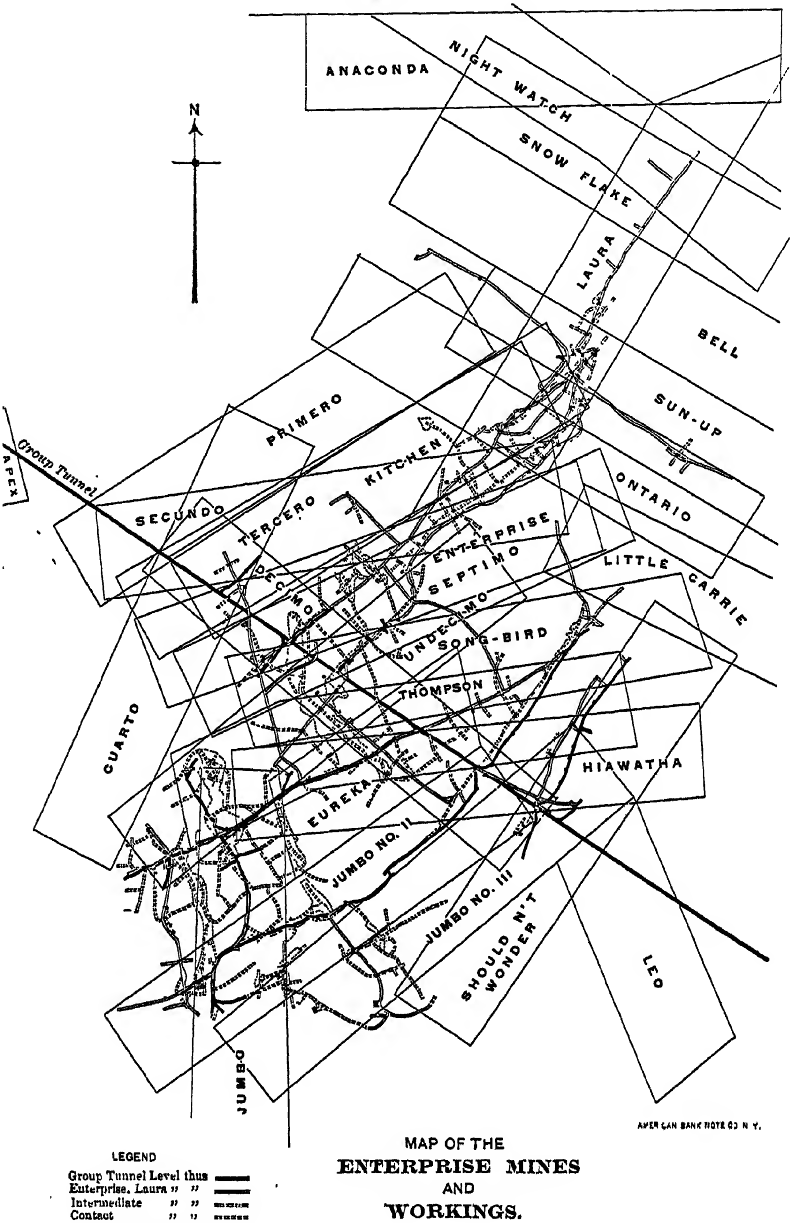
at Rico, and a second Leadville uncovered. A "rush" set in. In the fall of that year Messrs. Jones and Mackay, of Comstock fame, visited the camp and purchased the Grand View group of mines. Next year, 1880, the boom continued, and the erection of a smelter* was begun. The material required for construction all came on mule-back over the ranges from Alamosa at a cost of 16 cents per pound. In the fall the furnaces were blown in under the superintendence of Messrs. Endlich and Arnold.

All the early discoveries of this district centered around Nigger Baby hill and the valley at its base. In 1879, however, a shipment was made to Swansea from a discovery by Harry Irving on a claim located further south, on Newman hill, which is like a footstool to Mount Dolores. This event, unimportant as it seemed at the time, marked the beginning of the development which more than ten years later led to the prolific production of gold and silver out of the workings of the Enterprise and Rico-Aspen mines.

In the spring of 1881 David Swickhimer, Patrick Cain and John Gault sank a shaft 35 feet deep upon their Enterprise claim on Newman hill. This work was undertaken not upon the evidence of ore, but in the expectation of cutting the continuation of the veins successfully worked in certain claims further south, owned by the Swansea Gold and Silver Mining Company. Without entering into a detailed description of the geological structure of Newman hill it is necessary, in order to make the early story of discovery clear to the reader, to say that the true rock (sandstone and limestone) is overlain by drift, through which shafts must penetrate before reaching the ore-bearing formation. The veins do not reach the present surface, save in the face of the landslip where Harry Irving first detected them. The three owners above mentioned traded their claim to George S. Barlow for $300 worth of lumber. Barlow continued the sinking of the shaft to a depth of 146 feet. On an adjoining claim, named the Songbird, another miner, A. A. Waggener, sank a shaft to the depth of 203 feet. The latter penetrated through the drift into lime shale; but the En-

* That smelter still exists. It has afforded many well-known metallurgists their early and hard-bought experience. Its history would present an amusing commentary on the struggles of ill-digested enterprises.

Fig. B.



E. W. Hunt, Surveyor.

terprise shaft did not at that time reach the true rock. Both shafts got into very wet ground. In the meantime the Swansea workings were reported to be impoverished and, finally, exhausted of ore. It was also said that the veins did not extend northward, but the real fact was that cross-veins had faulted the ore-bearing veins in a manner to be rendered clear later on in this account. Newman hill was discredited, and early in 1883 the Enterprise and Songbird shafts were abandoned.

A year later, Larned and Hackett resumed work in the Swansea levels, and, by mere accident, discovered that the veins had not come to an end, but were simply dislocated. They prosecuted development, proved the continuity of the ore and made large shipments. Their success induced Waggener and Barlow to relocate their abandoned claims late in 1886. But neither of them had any capital, and they were unable to overcome the heavy flow of water. In December, 1886, David Swickhimer bought out Waggener's interest, acting on knowledge obtained while working in the Swansea mine, which had satisfied him that the veins must extend into the Enterprise and Songbird claims. In March, 1887, he recommenced the sinking of the Enterprise shaft. In May he acquired one-half of Barlow's interest. In July the windlass was replaced with a steam-engine and a pump. All this time Larned and Hackett were drifting rapidly northward and threatened soon to reach the boundary separating their territory from that of Swickhimer and Barlow. Unless the two latter succeeded soon in finding a vein in place, so as to permit a valid location, the claims could be successfully disputed.* They therefore hurried the sinking, and in spite of bad luck, floods of water and a general lack of experience, they struck ore on the 6th of October at a depth of 262 feet. The first assay gave 2.1 ounces of gold and 519.4 ounces of silver per ton.

This ore was one foot thick and formed part of a "flat lode." In the light of later developments, this discovery is known to have been a piece of particular good fortune, for the maps of to-day prove that it was the edge of the biggest ore-body ever found on Newman hill, and that a shaft put down 20 feet further east would have missed it. This was the first evidence of the existence of a flat ore-deposit. Swickhimer thought at

* A good example of the iniquitous operation of our absurd mining law.

first that it was merely a roll in the Enterprise, an almost vertical vein. It was, however, soon proved by the workings to be a bedded formation, conformable to the enclosing country. The shaft was sunk 60 feet below this "contact," and a drift was run westward until the increased seepage of water, in the following spring, proved too much for the pump, and caused work to be confined to the contact. In July the water diminished, drifting was resumed, and in August, at a distance of 118 feet southwest of the shaft, the Enterprise vein was at last intercepted. The ore was 20 inches thick and assayed 3.2 ounces of gold and 285.5 ounces of silver per ton.

In May, 1890, the Songbird and Enterprise mines, together with much adjoining property, were acquired by the Enterprise Mining Company, the operations of which were directed by the writer from March 1, 1894, to February 28, 1895.

Fig. A shows the group of claims forming the property, and the railway connecting them with the town of Rico. The mine map (Fig. B) indicates the complex of drifts and crosscuts which follows the ramification of veins. The workings aggregate 8 miles in length. They have yielded ore whose gross value exceeds $3,500,000. Of this nearly one-quarter has been gold, the remainder silver.*

Entrance to the mine is made by the Group tunnel which has a course S. 56° 58′ E., and consequently cuts the ore-bearing veins almost at right angles. Moreover, since its line corresponds closely to the strike of the country, it intercepts the veins at an approximately equal depth below the contact. It would do so without variation, but for the step-faulting which accompanies the vein-structure.

The tunnel or adit is 2920 feet long. Near the entrance the

* The analysis of representative lots of ore gave the following results:

	First class.	Second class.
SiO_2,	29.2 per cent.	50 to 55 per cent.
Mn,	2.0 "	6 to 10 "
Fe,	11.8 "	6 to 10 "
Zn,	12.0 "	5 to 7 "
Pb,	10.2 "	2 to 3 "
S,	11.6 "	5 to 8 "
Au,	0.87 oz. per ton.	0.3 to 0.5 oz. per ton.
Ag,	221.50 " "	45 to 75 " "

The first class was mostly contact-ore, while the second class consisted of the bulk of the product of the verticals.

contact is 210 feet overhead, at the breast it has approached to within 35 feet. The largest drop is due to a down-throw on the so-called Leo cross-vein.

II.—The Country-Rock.

The Dolores, as it flows southward from the town of Rico, is overlooked on the west by Mount Expectation, and on the east by Newman hill. The river has eroded its own way and does not follow the line of a fault. The Lower Carboniferous beds, which form both its bed and the immediately flanking hillsides, can be traced across the valley. On Newman hill they are for the most part hidden by a deposit of Quaternary drift, the maximum thickness of which is about 400 feet, diminishing southwestward. The underlying shales, limestones and sandstones contain fossils which determine their stratigraphical place. The intrusions of porphyrite,* both plentiful and irregular in form, particularly at the northern end of Newman hill, afford an explanation of the metamorphism of the sedimentary rocks.

The country enclosing the ore-deposits consists of these shales, limestones and sandstones, having a strike N. 20° W. and an average dip of 10°. They are thinly bedded. Single beds are not extensive, one layer dwindling in thickness until it dovetails into another. Without necessary variation in width, the composition may change so that lime graduates into sandstone. These facts indicate that the sediments were laid down in estuaries and in such shallow reaches of water as permitted of swift changes in the conditions of sedimentation. The fossil remnants are of a kind that accords with this view.

The foregoing description applies especially to that portion (about 200 feet thick) of the formation to which the mine-

* The following notes on a thin section of this rock, a hornblende-augite porphyrite, were given by Mr. R. C. Hills, geologist of the Colorado Fuel and Iron Company.

Macroscopic Character.—The rock is grayish in color, and shows white, opaque feldspar (plagioclase), evidently much kaolinized; also small, partly-altered green hornblendes. Apatites are occasionally visible under the lens.

Microscopic Character.—Under the microscope the feldspars are seen to be largely altered to kaolin. So far as determinable in the only section available, they are plagioclase. The green hornblendes are largely altered to chlorite. Small pale-green augites and stout, relatively large apatites are rather numerous, together with ore particles (magnetite). The granular groundmass is much kaolinized, and abundantly distributed through it are grains and microlithic crystals of feldspar, also kaolinized.

workings are practically confined. The veins do not penetrate upwards beyond the horizon known as the "contact," and they become barren at an average depth of about 150 feet below that horizon.* For this reason the overlying rock has been merely penetrated in sinking to the ore-bearing horizon, and, similarly, the underlying beds have only been pierced by one or two unsuccessful shafts and bore-holes.

The beds above the contact consist, in ascending order, of:

2 to 5 ft. of lime breccia;

A thin bed of soft, crushed sandstone, which rarely reaches a thickness of 9 ft., averages less than 1ft., and is occasionally entirely absent;

6 to 8 ft. of black shale;

30 to 40 ft. of sandstone beds, and

40 to 50 ft. of black lime-shales.

The last graduate into a series of blocky limestones, the escarpments of which appear on the face of Mount Dolores. In none of the beds of this series have profitable ore-deposits been found, although large veins of calc-spar traverse them at intervals.

The contact is not an ore-measure lying between two persistent beds of shale and limestone, as has been stated.† The composition of the encasing rocks is variable because of the comparatively brief persistence of individual members of the sedimentary series. It may be said, however, that the ore of the contact is invariably found in rock which has undergone shattering. An appearance of undisturbed solidity is occasionally given by later cementation. Several raises put up to the contact from the upper main level of the mine afford sections of the formation. Two are quoted.

No. 1.	No. 2.
10 in. of compact pulverulent, lime,	2 ft. of brown lime breccia,
14 in. of lime breccia,	6 in. of fine-grained sandstone.
2 in. black shale,	21 in. dark blocky limestone,
19 in. limestone,	3 in. black soft shale,
12 in. sandy limestone,	28 in. blocky lime,
10 in. sandstone,	5 in. laminated sandstone,

* See Figures D and E, p. 974.

† J. B. Farish, *Proceedings Colorado Scientific Society*, vol. iv., p. 154.

No. 1.

1 in. parting of black mud,
2 ft. lime shale,
8 in. crushed lime,
25 in. blocky limestone,
10 in. soft sandstone,
1 in. parting of shale,
Then a series of thin beds of sandstone aggregating 21 ft.
18 in. light-colored limestone,
10 ft. sandstone,

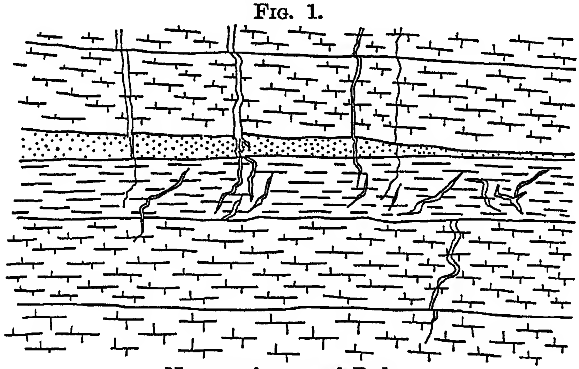
No. 2.

18 in. black shale,
1 in. parting black mud,
15 in. light gray limestone,
1 in. parting,
58 ft. sandstone,
$2\frac{1}{2}$ ft. shale,
7 in. sandstone,
3 in. shale,
$4\frac{1}{2}$ ft. sandstone,
2 in. lime shale,
3 ft. coarse sandstone.

Then a further series of sandstones.

The section given in the first column came from the first raise on the Jumbo No. 3 upper level, the second from a raise

Fig. 1.



Non-persistence of Beds.

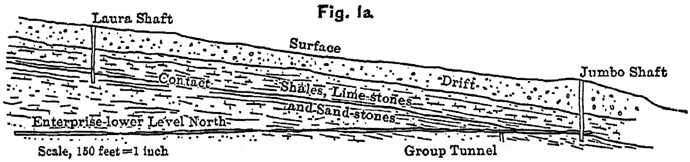
on the Kitchen vein from the Enterprise level at the end of the cross-cut between raises 7 and 8, Songbird. The linear distance between the two sections is only 900 feet. Beyond the similarity of the breccia, which marks the "contact," these two sections are entirely dissimilar, and it seems impossible to recognize any continuity in the stratification. The comparison serves to explain the statement already made that the beds of the series are notably non-persistent. The sketch reproduced in Fig. 1 represents the face of a cross-cut where a bed of sandstone has been caught in the act, as it were, of merging into a bed of lime-shale.

Further sections of the contact-horizon will be given when we come to consider the ore-distribution at that level.

It has been seen that below the contact comes a series of very thin lime and shale beds, interrupted by occasional sandstone divisions. These beds are all dark in color, graduating from coal-blackness at the contact to dark grays at a distance of 100 or 150 feet from it. As the contact is left, the sandstone beds become more frequent, their grain is notably coarser, the limestones become less shaly and more blocky, the black shale is absent, and soon the workings penetrate into thick beds of a coarse, light-colored quartzitic sandstone.

One or two shafts have been sunk, but the records which have been kept are unfortunately so vague as to be useless for the purposes of a geological section. The Jumbo shaft penetrated 524 feet below the contact, without any discovery of importance. The Skeptical shaft, just north of the Enterprise property, was sunk 365 feet in porphyrite and then penetrated 15 to 20 feet of shales and limestone. A bore-hole subse-

Fig. Ia.



SECTION THROUGH JUMBO AND LAURA SHAFTS.

quently put down in the bottom of this shaft went through 200 feet of shales and limestone before entering quartzite, where operations ceased. The Lexington tunnel, which is 400 feet below the contact, penetrates the Newman hill formation for a distance of 2740 feet, and is in coarse, light-colored, hard sandstone for most of its length.

There has always been much surmise regarding a certain undiscovered "second contact." It has been the cause of much nonsensical mining. The Enterprise Company, in 1893, sunk the Jumbo shaft in search of this lower ore-measure, not realizing that, owing to the position of the shaft and the dip of the formation, they would have to go down 300 feet below their main adit before they would be even level, in a geological sense, with existing northern workings.

This piece of exploratory work was badly planned and proved without result. The neighboring company, the Rico-Aspen Consolidated Mining Co., put down a bore-hole, which was as-

barren of encouragement. The "second contact" of Newman hill is a vain imagination. A hazy idea of the geology of Leadville, blended with a misconception of that of Rico, has caused the growth of an idea having no facts for its support. It was suggested by the occurrence, on the hills north of the town (above the village of Piedmont), of a series of at least three ore-bearing contacts, and it seemed to be indicated by the developments in the Atlantic Cable mine. But deductions from this evidence are vitiated by reason of the fact that the valley of the Dolores, near the north of the town, is crossed by a large dike of porphyrite, marking a fault which breaks the continuity of the country on either side.

The Atlantic Cable Co.'s bore-hole gave the following downward section:

	Feet.
Limestone,	7
Lead- and zinc-ore,	4
Limestone,	5½
Lead- and zinc-ore,	5
Limestone,	13
White marble,	20
Zinc-blende ore,	3
Specular iron-ore,	18
Limestone,	43
Porphyrite,	1
Limestone,	25
Porphyrite,	2
Limestone,	3
Mineralized porphyrite,	3
Porphyrite,.	21

The remaining 170 feet of the hole continued in quartzite. To render the evidence complete, I now append the record of the Rico-Aspen Co.'s bore-hole. This was sunk 20 feet northwest of the Jumbo vein, between raises 7 and 8, at a point 85 feet below the contact. It was begun February 12, and finished September 10, 1895. From the collar to a depth of 481 feet the drill traversed alternating beds of limestone and sandstone; the latter becoming coarser as depth was attained. From 481 to 541 feet the drill traversed porphyrite. Between 541 and 573 feet the rock was quartzite. Then porphyrite continued to the bottom of the hole, at 706 feet. The evidence afforded by these borings w'll be referred to after other matters have been passed in review.

III.—The Ore-Occurrence.

In the investigation of the relation between the ore-occurrence and the rock-structure it is found that there are two distinct systems of vein-fissuring. One series of veins has a N.E.–S.W. strike and a nearly vertical dip; and this series is crossed and faulted by a second system, having an approximately N. and S. trend and a flat dip. The former are ore-bearing and are called "verticals" or "pay-veins;" the latter are barren of valuable ore and are termed "cross-veins." Both series fail to reach the present surface, save where deep erosion has occurred, because, in coming up through the Carboniferous formation, they are abruptly terminated in their near approach to a certain horizon marked by the occurrence of black shale and beds of crushed lime. This, the contact, is disturbed by the cross-veins.

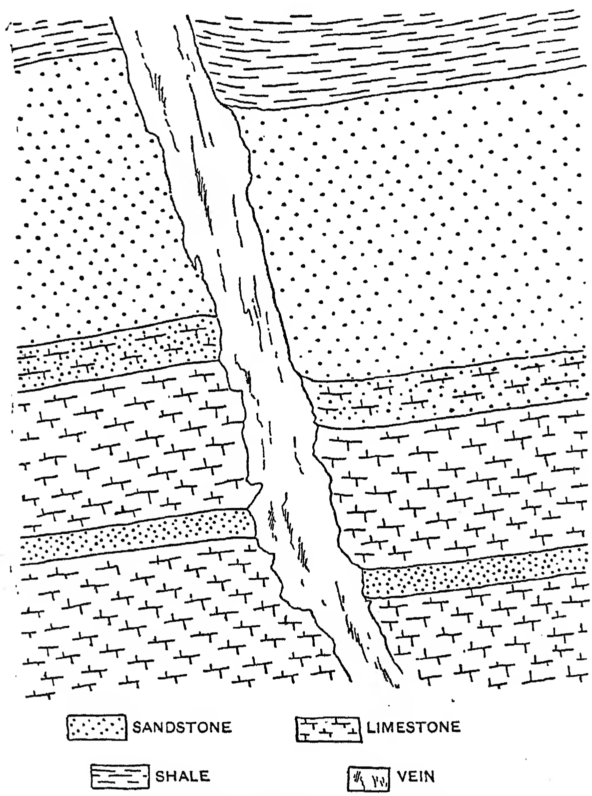
The verticals are not productive immediately under the contact. On the contrary, when within a distance varying from 5 to 15 feet from that contact they split up into stringers, which scatter the ore so as to render exploitation unprofitable. Apart from this dispersion of the vein, the total amount of ore which it carries is also decidedly lessened. The plane of the contact is itself ore-bearing; the bodies occurring in the form of narrow channels ramifying through the crushed rock, in directions which correspond exactly to the strike of the veins underneath. It is a notable fact, moreover, that the cross-veins, barren as they are, are yet related to ore-bodies on the contact as rich as, if not richer than, those above the line of the verticals.

The knowledge of the relationships just outlined has proved vital to the intelligent exploration of the mines; and the theoretical consideration of them is, to the student of ore-deposition, highly suggestive. The structure of the formation, on account of the narrowness of the veins and the thinness of the beds through which they pass, affords within the space of a few square feet sections which ordinarily it requires acres to encompass. The coloring, moreover, of the minerals accompanying the ore and of the rocks enclosing it, is so marked as to assist the ready interpretation of structure, and enables the observer to portray them by pen and pencil. For these reasons the writer has endeavored to give the testimony collected by him in the form of a series of drawings, rendering much comment unnecessary.

IV.—The "Verticals" or Ore-Bearing Veins.

These belong to the simplest type of ore-deposit. They are fractures cutting across the sedimentary rocks almost at right angles to the bedding-planes. They have a simple structure. Their width averages less than a foot; they are built along fault-lines, are sensitive to the changes in the encasing rock, and are themselves faulted by veins of later formation.

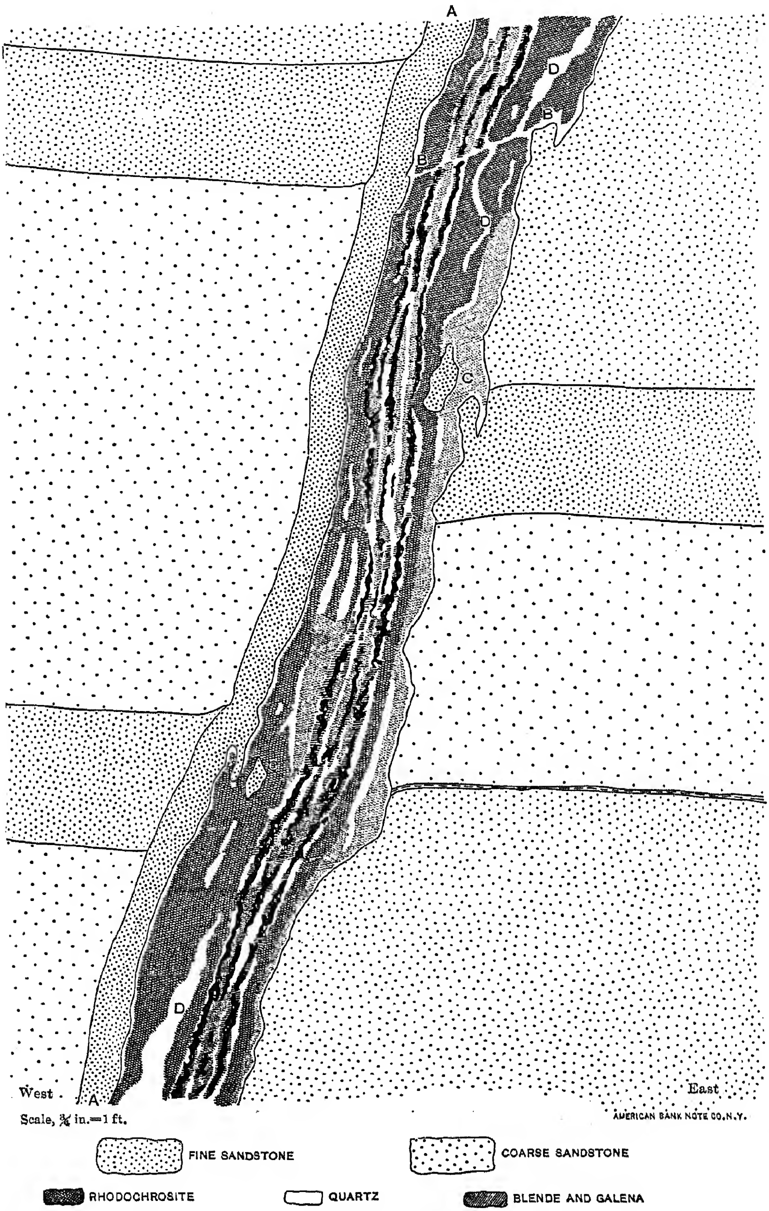
Fig. 2.



A Typical Pay-vein or Vertical.

About a dozen veins have undergone noteworthy development, and of these, five have yielded the bulk of the ore-production of the mine. The Enterprise, Jumbo No. 2, Jumbo No. 3, and Hiawatha all dip to the northwest at angles varying from 5° to 15° from the vertical; the Eureka is practically vertical; while the Kitchen, Swansea and Songbird veins have an

FIG. 3.



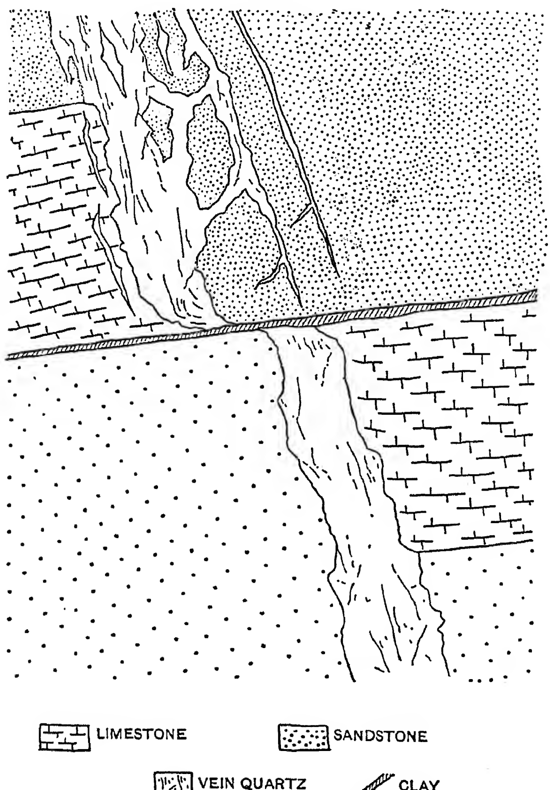
SONGBIRD VEIN

Scale, $\frac{3}{4}$-inch = 1 foot.

opposite (southeast) dip and a flatter angle, viz., from 12° to 22° from the vertical. Their strike varies between 50° and 65° east of north, the Eureka being conspicuous for its regularity.

Fig. 2 is a typical illustration obtained from the end of one of the levels. The vein is from 5 to 6 inches wide and cuts

FIG. 4.



Vertical and Horizontal Movements.

the country-bedding at a right angle. It will be noted that the ore occupies the line of a fault, the throw of which is 7 inches; the direction of the movement which caused it being indicated by the turning up of the bedding-planes on the hanging-wall of the vein, and a corresponding bend in the partings on the foot-wall side. The varied texture and color of the beds of sandstone and limestone rendered this structure very distinct.

Fig. 3 represents the Songbird vein, which has a dip op-

posite to that of the Jumbo No. 3, just described. The drawing came from a stope about 30 feet below the contact, where the vein happened to be entirely encased in beds of sandstone. A fault is evident. Its throw is about 28 inches. The ore is 9 to 12 inches wide. On the hanging there is a casing, A A, 3 to 4 inches thick, which follows the vein throughout the section. This casing, of dark sandstone, is separated by a slight selvage from the outer country, but graduates gently into the vein-stuff adjoining it on the east. On the foot-wall there is a marked selvage, accompanied by crushed rock. Two inclusions, C C, of sandstone occur within the ore. The latter is banded by streaks of zinc-blende and by ribbons of rhodochrosite within the quartz. D D is quartz. At B B the ore has been slightly dislocated.

In Fig. 4 more complex conditions are represented. Two dislocations of the country are evident. The vertical break followed by the ore-formation has a throw of about 2 feet, while the lateral fault, evidently of later occurrence, has caused a disturbance measured by 9 inches only. It will be noticed that the parting separating the upper fine-grained sandstone from the bed of limestone has been brought into line with that dividing the same bed of lime from the underlying coarser sandstone. This coincidence must have facilitated the subsequent lateral shifting of the rocks. Such occurrences are frequently observable in the mine.

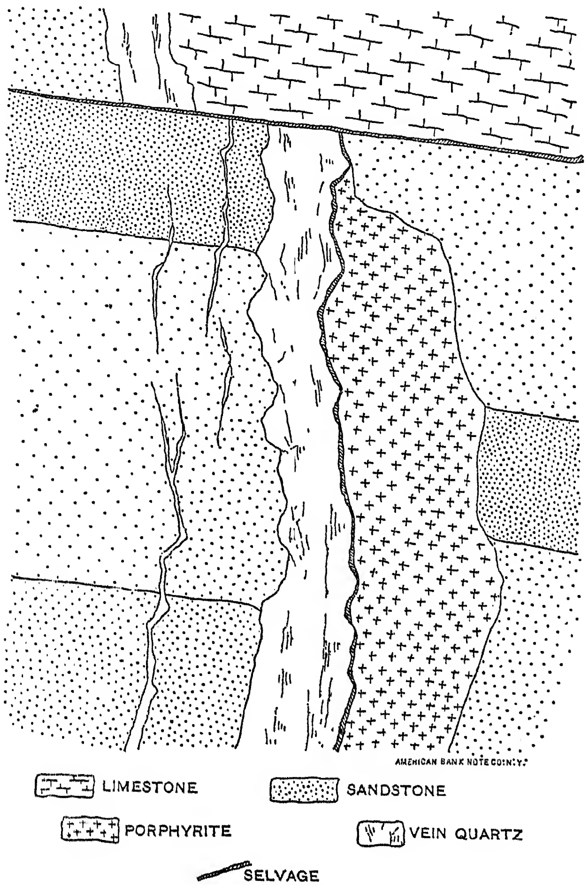
It may be questioned whether the movement along the bedding took place before or after the ore had been laid down. There is evidence elsewhere in the mine that such movements have both preceded and succeeded the vein-formation. In this case it preceded, because the ore is seen to be not abruptly broken off, but shaped to the structural conditions created at this point previous to its precipitation.

A different state of things is disclosed in Fig. 5, which represents the Jumbo No. 2 vein, as seen in the end of the lower level. Here, as usual, the ore occupies a fault-fracture, which has dislocated the bedding to the amount of 20 inches; but, in addition, a later movement along coincident partings has broken the vein and thrown it about a foot. The ore has been shattered, and in the clay accompanying the line of fault there are pieces of quartz and rhodochrosite, evidently due to this shat-

tering. The quartz-veins, unaccompanied by ore, observable to the left of the vein, are of later origin. The dike of porphyrite will be referred to elsewhere.

In Fig. 6 a similar later movement, but this time in a ver-

FIG. 5.



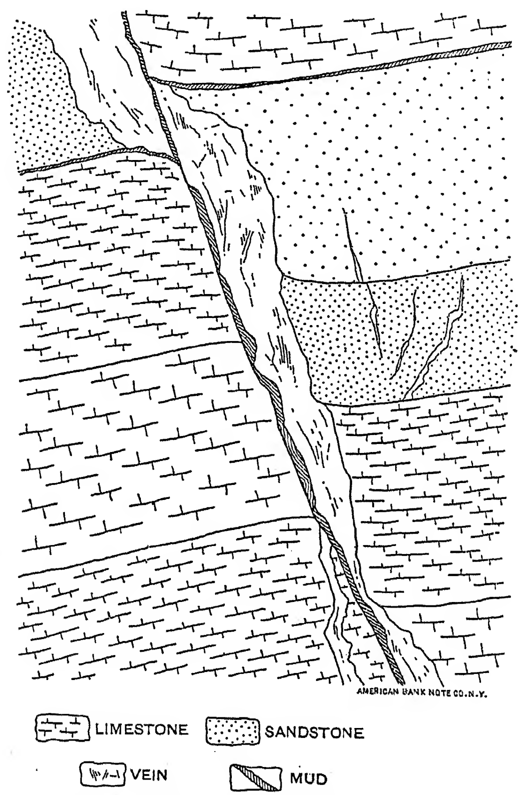
Fault along Bedding-plane.

tical direction, is illustrated. The Jumbo No. 3 vein, here shown, follows a fault the throw of which is about 2 feet. Since the ore was laid down a later shifting of the country has been accompanied by the formation of a fracture, which approximately follows the line of weakness of the older movement, and breaks across the ore lying in its path. The amount

of this dislocation cannot be measured with certainty; it is probably slight.

The veins are, as to their size, behavior and ore-bearing character, very sensitive to the structure of the enclosing rock. They flatten when traversing lime, the increased deviation from the vertical being accompanied by a diminution of ore.

Fig. 6.



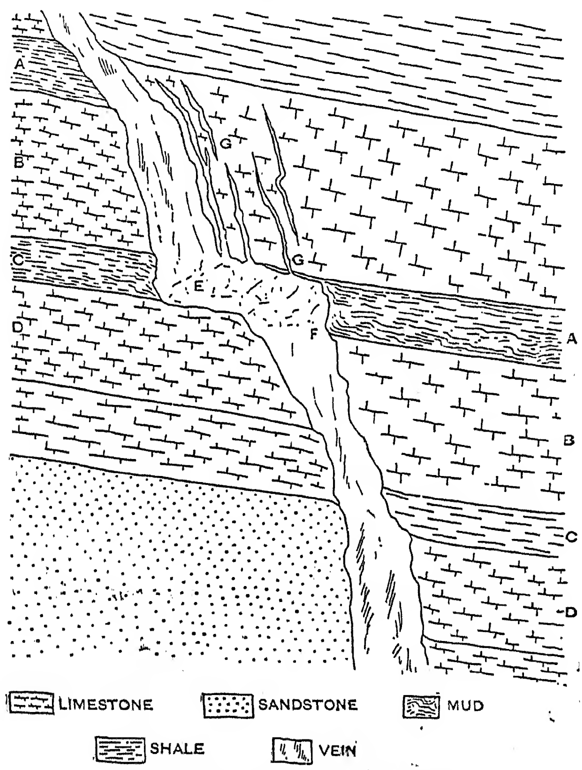
Later Movement along Original Fracture.

Even in those cases where the actual width may not decrease the percentage of valuable minerals does. In sandstone they usually straighten up, and are marked by an enrichment. When crossing a parting between the beds an offset of ore is often formed underneath the parting. Some of these characteristics are illustrated in Figs. 7 and 8, both representing the Enterprise vein. In both the faulting along the ore-break can

be measured, since the dislocated portions of the same bed are similarly lettered.

Among the miners it was the common saying that "the vein makes ore in sandstone," but my observations did not quite confirm this generalization. When traversing lime the veins tend to split up into stringers, and this is the case, to a lesser

FIG. 7.



Vein Flattening along a Shale-bed.

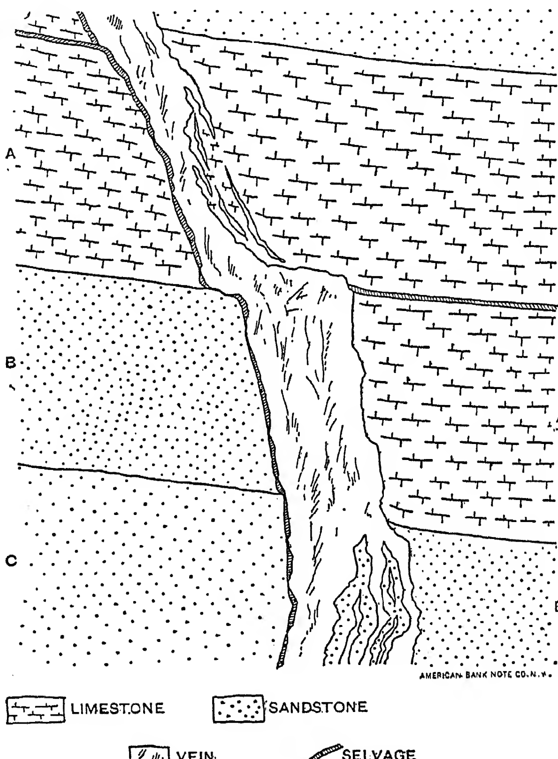
degree perhaps, in sandstone. On the whole, my experience was that unlike walls give the best environment for rich ore, and that a foot-wall of sandstone with a hanging of lime is a particularly favorable combination.

In some cases comparatively modern shiftings of the country are evidenced. Thus in Fig. 7 the flattened part of the vein, E F, formed along the shale band, is crushed, and it is my belief that this was caused by a movement along the bedding

long subsequent to the formation of the vein itself. The stringers, G G, are also of late origin, and are composed of barren quartz dissimilar to the gangue. In Fig. 8 there is a similar jog in the vein, but in this case no crushing or disturbance is suggested.

Pronounced selvages are not characteristic of these veins.

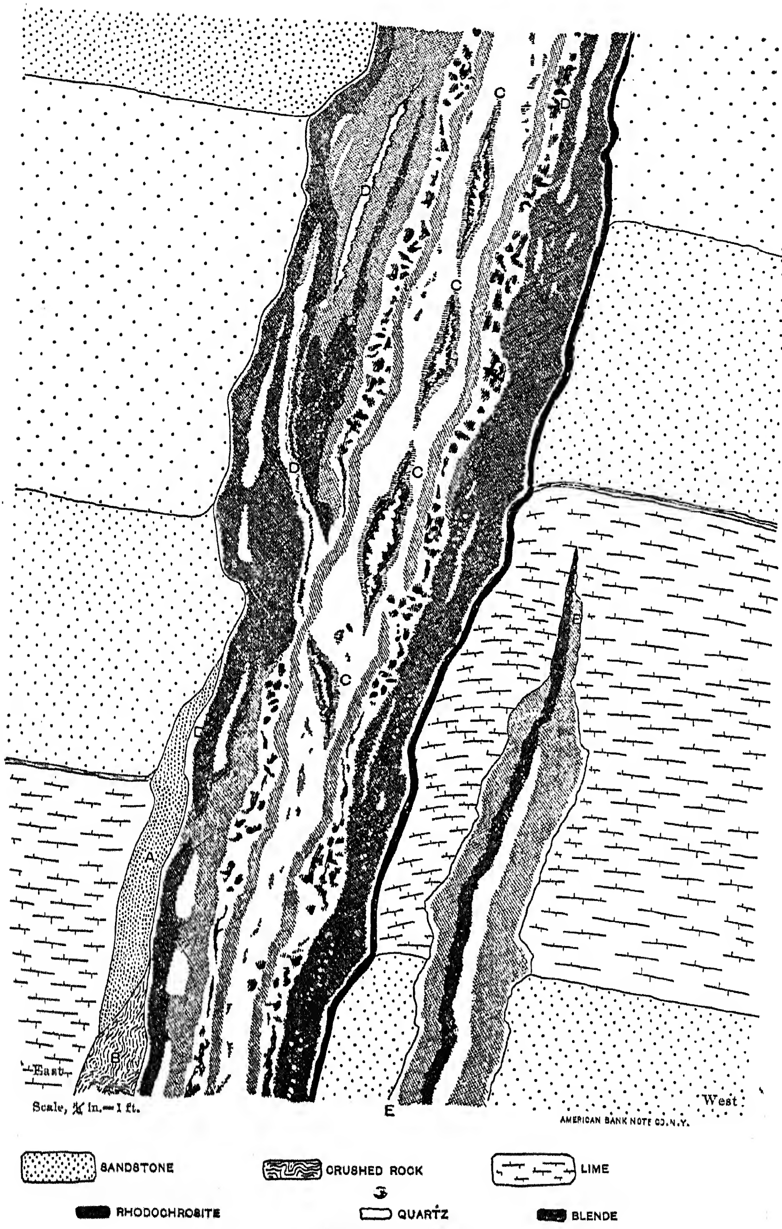
FIG. 8.



Influence of Encasing Rock upon Behavior of Vein.

When noticeable they are usually in lime-beds. In sandstone they are comparatively infrequent, and the absence of a parting causes the ore to separate with difficulty from the sandstone, into which it merges in such a manner as to cause the miners to say that it is "frozen" to it. A casing of sandstone is occasionally seen when the vein is entirely enclosed by lime, proving that the sandstone must have been shorn off upper beds in the course of that movement which determined the existence

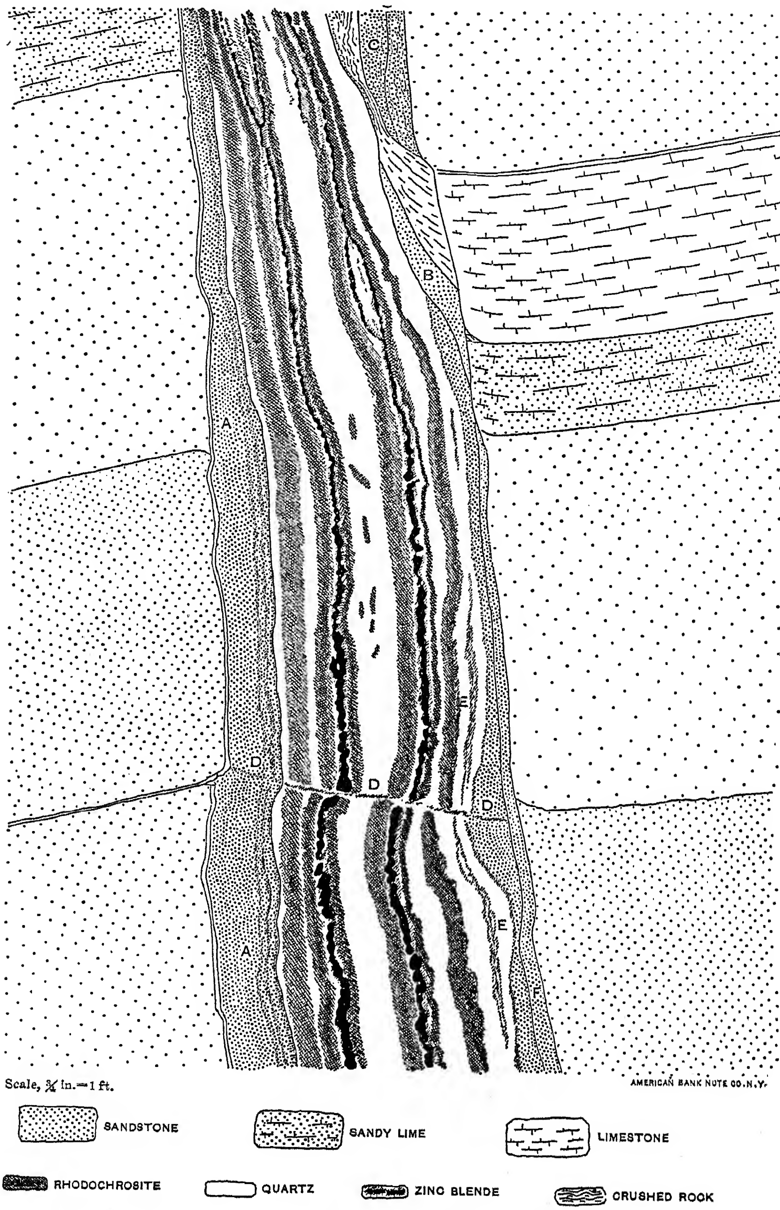
FIG. 9.



JUMBO No. 3 VEIN

Scale, $\frac{3}{4}$-inch = 1 foot.

Fig. 10.



EUREKA VEIN

Scale, ¼-inch = 1 foot.

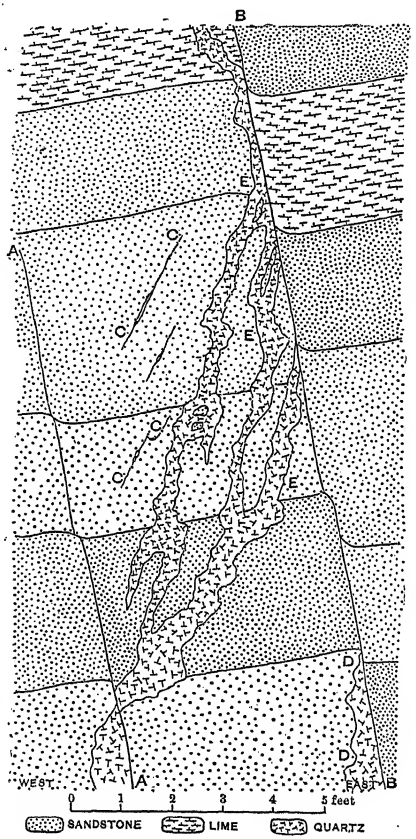
of the vein. This feature is illustrated in Figs. 9 and 10. The first of these represents the Jumbo No. 3 vein, which in this instance is identified with a fault of about 3 feet throw. On the foot-wall the ore is divided from the country by a clay selvage, but on the hanging there is no such parting. Toward the bottom of the section the vein exhibits, on its hanging-wall, a casing of country which changes from sandstone (A) to lime shale (B) in accordance with the succession of similar adjacent beds. The ore is distinctly ribboned by a symmetric alternation of vari-colored minerals. D is quartz. The vein is bilaterally symmetrical on either side of a central line marked by vugs or cavities, C C, which are encrusted with quartz crystals. The blende on each side of these vugs occurs in a curiously spotty manner, suggesting brecciation. The smaller vein, E E, consists of barren quartz and rhodochrosite. It is of apparently later origin.

In Fig. 10 we have a very striking example of this ribboned structure, to be discussed later on. For the present attention is directed to the sandstone casings which follow the walls of the vein. That on the hanging (F F) is scarcely an inch wide, and has no appreciable selvage separating it from the outer country. That on the foot-wall, A A, varies in thickness from 1½ to 2¾ inches, has a distinct parting dividing it from the country, and is, moreover, marked by a dark streakiness, suggesting incipient ore-formation. In both cases these casings graduate gently into the adjacent vein-stuff.

Each vein follows a fault-fracture. The shifting of the country is not likely to have been limited to a single line of faulting, and it is found that just as the series of "verticals" indicate contemporaneous and approximately parallel movements, so there are also other minor shiftings sympathetic to these, unaccompanied, it may be, by ore, and therefore unexplored by the miner. Such subordinate faults are occasionally seen close to the vein. Fig. 11 is a section of the Jumbo No. 3 in a stope where it was non-productive, being represented merely by a barren quartz vein, carrying a little rhodochrosite, but no valuable sulphides. Two lines of faulting, A A and B B, are evident. The vein first follows one of these fault-planes and then deviates along cross-joints until it meets the other, 5 feet further west, which it then accompanies. Both fault-lines are marked by a selvage. The ore lies on the under side. The

main lode (DD) reappears (along BB) $5\frac{1}{2}$ feet lower down. Deeper still it crosses over to the western fault, as its branch had previously done, and, uniting with the latter, forms a strong, rich vein, which continues undisturbed to a further depth of 30 feet, when another irregularity breaks its continuity.

FIG. 11.

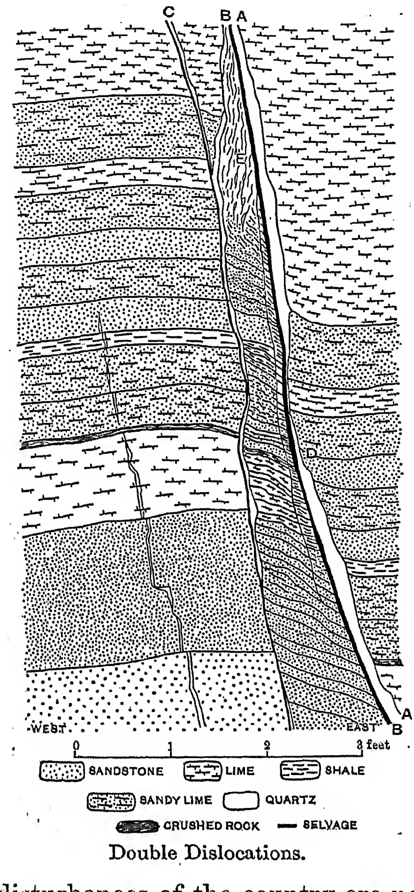


Vein Changing from one Fault-plane to Another.

Fig. 12 affords further evidence. In this case, as in the last, the vein is seen where it is small and poor. Under such conditions its structure is more readily discernible, because enrichment and enlargement generally produce confused outlines and are accompanied by a generous mineralization, destructive of

definition. Hence the stopes most instructive to the scientific investigator are least pleasing to the mine manager. In this section there are two veins, both small, of which the western may be regarded as a mere off-shoot. Two dislocations and

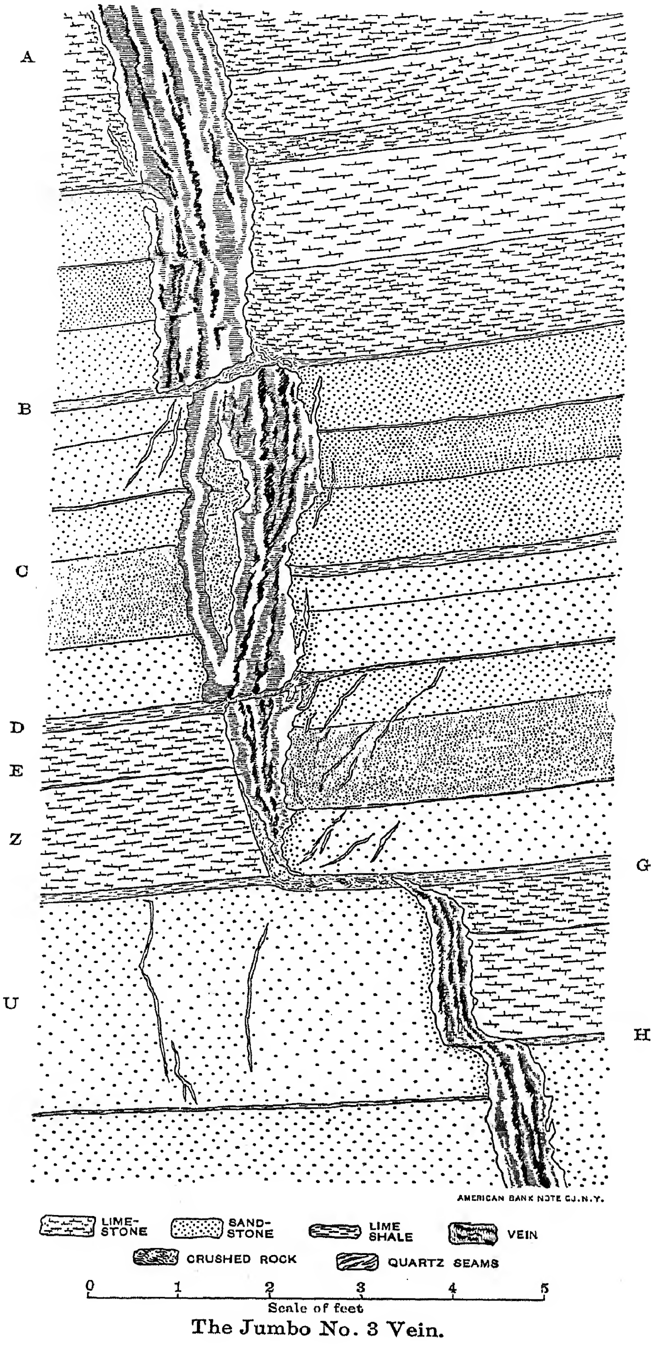
FIG. 12.



Double Dislocations.

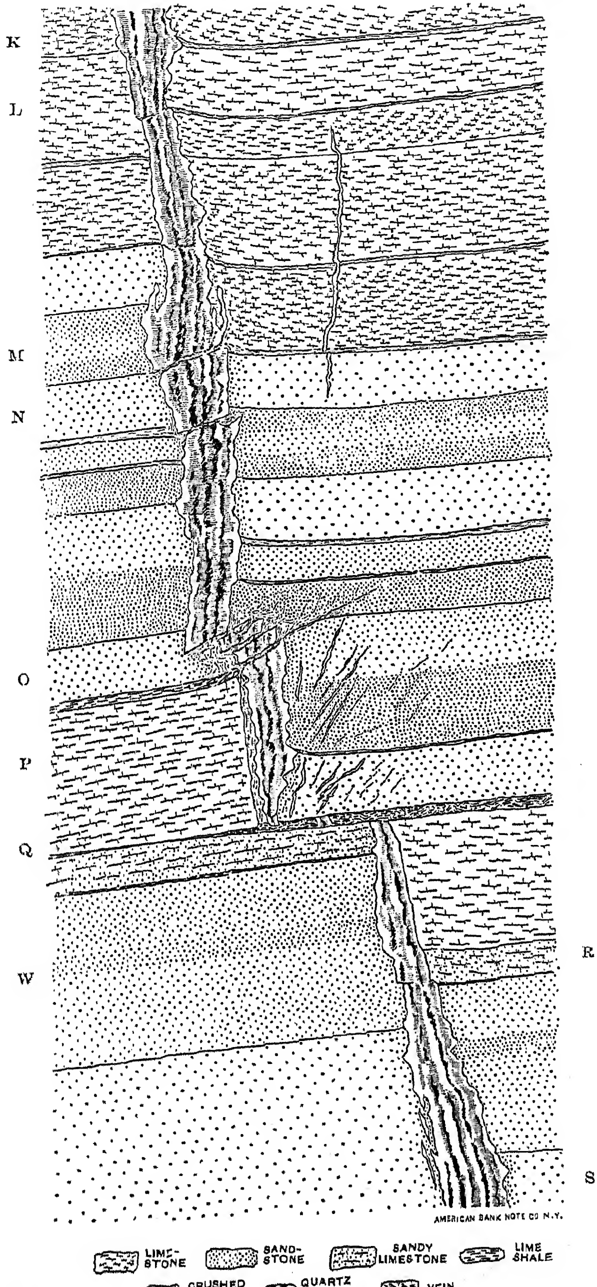
several minor disturbances of the country are noticeable. The western quartz-seam occupies a fault of a few inches, which dies out into a mere distortion of the bedding; the larger vein is identified with a fault having a throw of $2\frac{1}{4}$ feet. The bending of the edges of the beds as they abut against the vertical quartz-veins is very marked. Slight shiftings along bedding-planes are indicated by the behavior of the small stringers traversing the country.

Fig. 13.



The Jumbo No. 3 Vein.

Fig. 14.



The Jumbo No. 3 Vein.

The veins are built up of many-colored minerals, which give them a rare beauty, and serve also to accentuate their structure. Rhodochrosite and quartz enclose the sulphides of zinc, lead, iron, copper and silver in the form of galena, blende, iron and copper pyrites, argentite and stephanite. Native gold and native silver both occasionally accompany the argentite. A banded- or ribbon-structure is frequently brought about by the alternation of quartz, rhodochrosite and the sulphides. This structure assists the breaking of the ore, which will often part in ribbons within itself more readily than it can be detached from the encasing country. Fig. 25 illustrates a typical piece of vein-stone.* The gentle graduation of ore into country is noteworthy. The banding due to the rhodochrosite is a distinguishing feature, while the inclusion of portions of sandstone and the comb-structure of the quartz are additional testimony as to the origin of the ore, which will be discussed under another heading.

Much more might be said concerning the behavior of these veins; but sketches of the actual occurrences are better than verbal description. Fig. 13 shows the Jumbo No. 3 lode, as seen in the stopes of the mine. The vein structure is illustrated for a height of sixteen feet. It will be observed that the vein follows a line of fracture which has faulted the country. The vertical dislocation measures 2½ feet, and is rendered easily evident by the partings of shale which separate alternating sandstone and limestone-beds. Following the section downward, the ore is about 14 inches wide at the top, opposite A, and is distinctly ribboned with bands of quartz, rhodochrosite and zinc-blende, the last being also mixed with galena. The vein continues fairly uniform for five feet, and is then interrupted by a break opposite B, which indicates that the country has been shifted to the right for a distance of six inches. This movement took place at a point where a coincidence occurs between the partings between two sandstone-beds on one side and lime-sandstone-beds on the other. Below this point the vein opposite C is less regular, and divides into two branches, of which the eastern carries all the pay ore, the western being merely rhodochrosite and quartz. The latter, from its compo-

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* See this figure, and a discussion of it, at page 224 of the present volume, in the author's paper on "Vein-Walls."

FIG. 15.

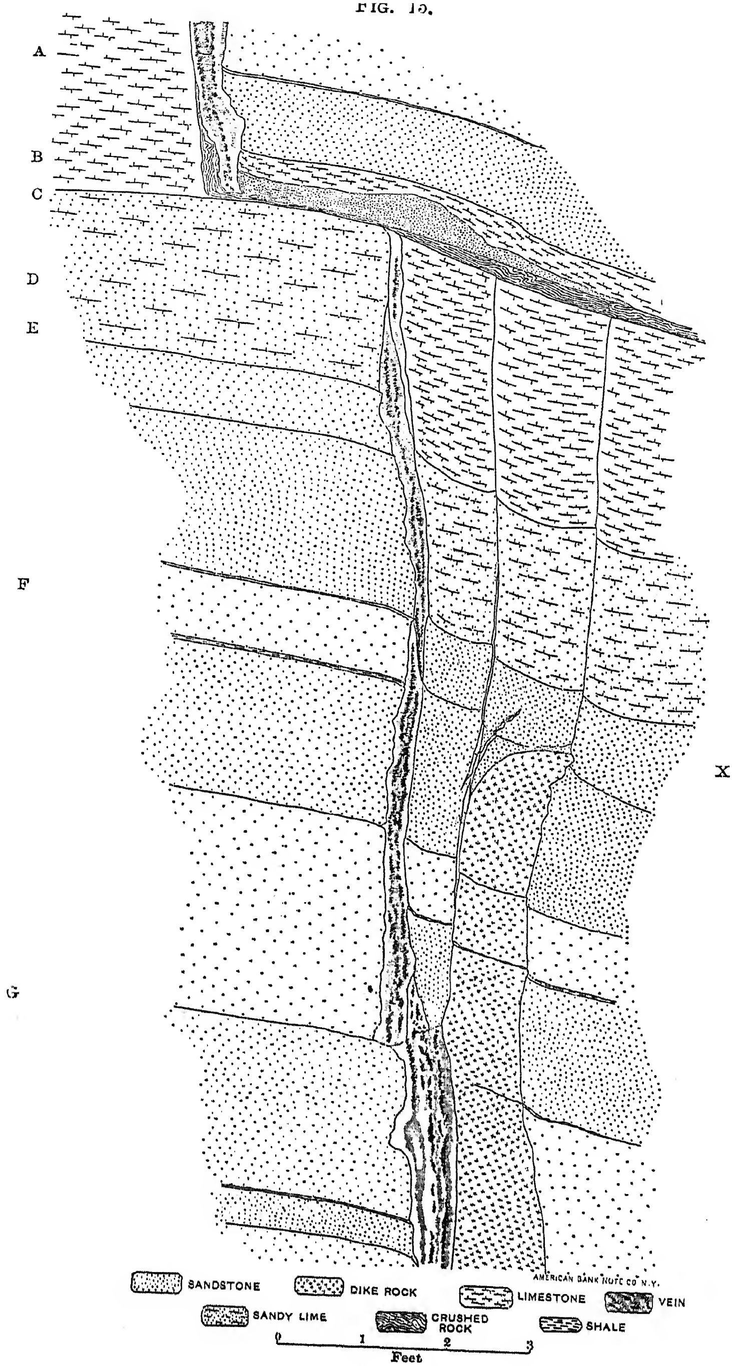
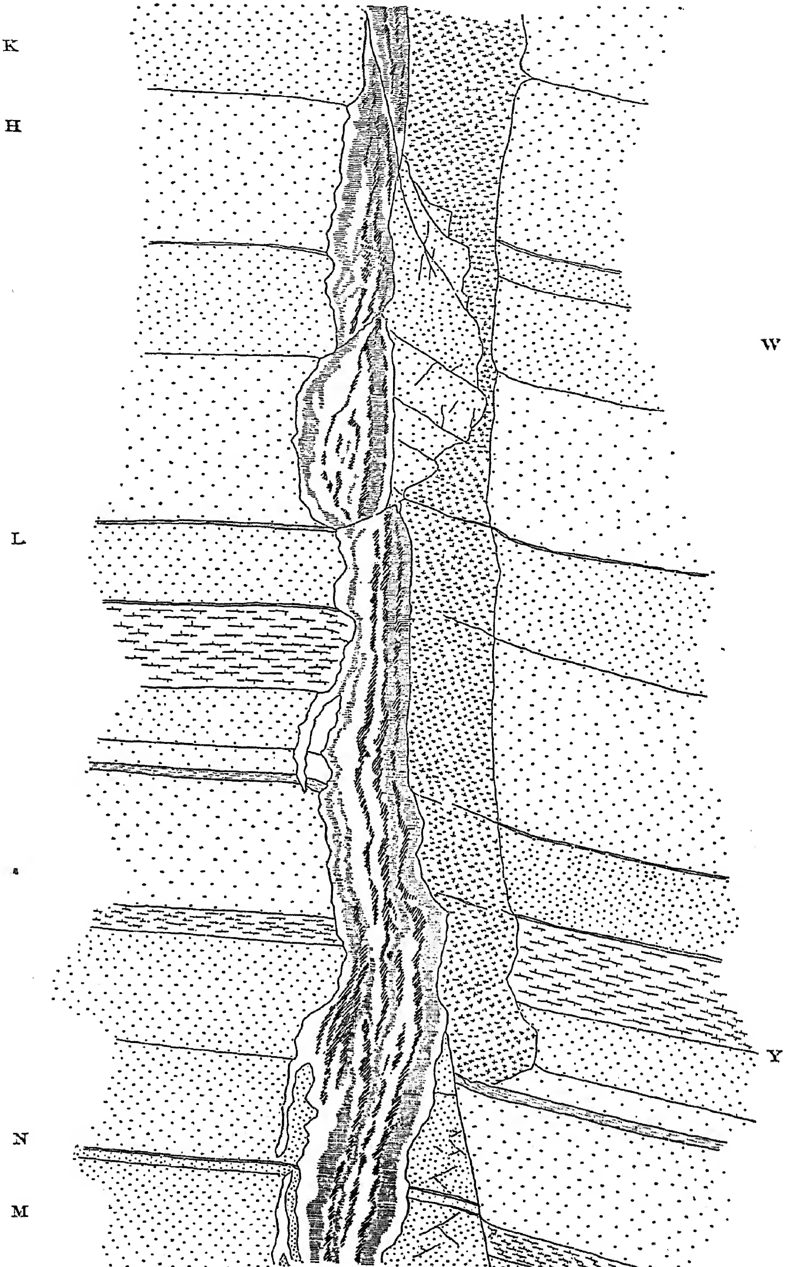

A
B
C
D
E
F
G
X
AMERICAN BANK NOTE CO N.Y.
SANDSTONE
DIKE ROCK
LIMESTONE
VEIN
SANDY LIME
CRUSHED ROCK
SHALE
0 1 2 3
Feet

FIG. 16.



Downward Continuation of Fig. 15, Constituting Section of Jumbo No. 3 Vein through Successive Stopes.

sition and structure, suggested to me, at the time, that it was of later origin than the ore-bearing vein. Opposite D, four feet below the last break, another occurs, a clean transverse fracture, accompanied by scarcely any shifting. This, too, is found where partings on opposite sides are so placed as to very nearly make a continuous plane. The vein has been slowly diminishing, and opposite E is only 8 inches wide. It continues to dwindle. Independent little quartz stringers to the right suggest disturbed ground. The vein decreases to a mere thread and ends in a soft shale parting, G, two inches wide. But other more interesting features demand notice. The ore no longer clings to the left or western side of the fault-fracture, which determined the original vein-formation. Moreover, its course downward is abruptly terminated. Several fragments of ore in the shale band lead naturally to the continuation of the vein, discovered to be 18 inches to the east. Here, opposite G, the country has been shifted by a movement equal to about 1½ feet; but the behavior of the vein, especially the bending of the ore near the points of fracture, suggests that the dislocation of the country took place before the vein was formed. The broken fragments indicate later movement also. Two feet lower the vein is fractured and slightly dislocated. Below this point the ore widens to 8 inches, and so passes out of view.

The conditions represented in Fig. 14 existed in the stopes upon the same vein, about one hundred feet further north on the strike. It is particularly interesting as illustrating the changes not only in the vein but in the individual members of the series of sandstone, shale and limestone-beds through which the vein passes. Commencing at the top, K, of the section, the ore is seen to be seven inches wide. About 1½ feet lower, opposite L, there is an incipient break in the vein, not sufficient to part it. At five feet from the top, opposite M, a rupture has taken place, accompanied by only a very slight displacement. Eleven inches lower, opposite N, there is another break. M and N correspond to B in Fig. 13. The ore here is 9 to 10 inches wide. The vein is well defined and nearly vertical. Opposite O, three feet below N, there is a more serious break. The vein is displaced its own width, and for a height of six inches is broken into several definite fragments. This severe shattering is due to the fact that, in the absence of coincident partings, no clean horizontal shifting of the country took place, but the movement

became more of a distortion. Compare the corresponding break in Fig. 13, opposite D. The vein now decreases, as in Fig. 13, and leaves, opposite P, the foot-wall of the main fault-fracture. Opposite Q the ore is about $1\frac{1}{2}$ inches wide and two partings exactly coincide, so that the horizontal shifting of the country is expressed by a clean-cut fault in the vein amounting to a displacement of $1\frac{1}{2}$ feet. A piece of ore occurs in the clay-seam, midway between the divided parts of the vein. Below this the ore increases. Opposite R there is an incipient fracture, corresponding to H in Fig. 13. The ore then widens steadily, and, as it leaves the section at S, has a width of about one foot, neatly ribboned. In this section the vertical throw along the fault-fracture followed by the vein is a little less than in Fig. 13.

In Figs. 15 and 16 there is a representation of the same vein as seen in a series of six stopes covering a vertical height of 33 feet. These stopes were at least two hundred feet north of the place illustrated in Fig. 14, but in the same horizon of country-rock. The lime and sandstone at the top of Fig. 15 can be identified with the beds P, Q and W in Fig. 14, and with Z and U in Fig. 13.

At the top of Fig. 15, opposite A, the vein is only 5 inches wide, but is built up of rich sulphides, zinc-blende and galena. A little lower, opposite B, the ore leaves the foot-wall of the fault-fracture and then undergoes a dislocation of over two feet. This movement along the bedding of the country occurs where partings coincide. On the west the plane of movement, C, is not accompanied by much selvage, but eastward there is a seam of clay widening to 3 or 4 inches in thickness. Incidentally the want of correspondence in the composition of the country on the two sides of the vein attracts attention to the fact that it follows a fault-fracture, the vertical throw of which is three feet. Opposite D, one foot below the break, C, the ore exhibits what miners call a "splice," that is, one distinct band of ore thins out and another at the same time commences to appear. The vein opposite E is about six inches wide, and rich. Just above it was very poor and quartzose. Nothing noteworthy occurs until, three feet lower (F), another splice is seen. The vein is still small but fairly rich, and continues thus very uniformly for another five feet, when (G) several features attract attention. The vein appears fractured, although not separated, and abuts

against a dike of porphyrite nearly one foot wide, which forms the hanging-wall of the ore for a distance of five feet (G, Fig. 15, to H, Fig. 16). When traced up and down the series of stopes it was found to cease in both directions and to be therefore an intruding tongue. Its behavior is clearly shown in the drawing, beginning opposite X, Fig. 15, and ending opposite Y, Fig. 16. It varies little from a uniform width of slightly less than a foot, except for a short distance (H to W), where it is squeezed to a mere thread a few inches wide. The porphyrite is evidently very sensitive to the structure of the country-rock into which it has thrust itself. Opposite G the ore is about 7 inches wide. It continues so for four feet downward, when (K) another splice occurs and the vein widens. For the next six feet the ore varies from 6 to 12 inches, and is then (L) slightly dislocated. Below this the vein widens steadily and develops a beautiful ribbon-structure, passing out of the section (M) with a thickness of 14 inches. At the bottom of the section an instructive feature is presented. The ore-bearing vein leaves the main fault-fracture (here shown by a line to the right), and pitches slightly to the west. This is only temporary. Lower down, outside the section here shown, it turns back and resumes its course along the fracture which determined its existence. It will be noted that the ore in leaving the main fault-line (about opposite Y), follows a line of minor fracturing which shows a slight but evident dislocation also.

These four sections, Figs. 13 to 16, show how the behavior of the vein is determined by the structure of the enclosing rock. The fracturing through the country, which opened a possible channel for the circulation of the mineral-bearing waters, was continuous but irregular. The irregularity was due, in the first place, to the varying composition of the beds through which the fracture passed. That fracturing was accompanied, as we have seen, by a vertical displacement of from two to three feet. The direction of the throw is beautifully evidenced by the bent edges of the partings in the country on either side; on the hanging, upward; on the foot-wall, downward.

The horizontal shiftings which now break the vein occurred in part before ore-deposition, but mostly afterward. The later movements were also the parents of the quartz-stringers which

now fringe the vein, as at C and F in Fig. 13, and M and P in Fig. 14. They represent the healing-action of later solutions, circulating along a reopening of the old passage-ways.

The three sections, Figs. 13, 14 and 15, illustrating the same horizon in the country, at considerable distances apart, serve to prove how individual members of the sedimentary series change in composition. The two beds, E and Z, both limestone, in Fig. 13, become, in going northward, the two beds, P and Q, one limestone and one sandy lime. In Fig. 14 the relative thicknesses have changed. Again, in Fig. 15, the same beds, now seen 200 feet further north, are to be identified in the single bed of limestone extending from A to C. The underlying sandstone also shows variations at the different points. It is noteworthy, at the same time, that the vein acts much in the same way when traversing identical beds at different points.

The subsidiary fracturing, seen in Fig. 15, to the east of the vein, between C and G, is instructive. That this structure usually accompanies vein-faulting I certainly believe. The breast of a slope or the face of a drift rarely exhibits it, because mining does not require that the ground should be broken for a width necessary to make it visible. This structure, the sheeting of the country, is very marked at Cripple Creek, in Colorado, and is the origin of parallel and multiple veins.

V.—The Cross-Veins.

These veins, although non-productive, play an important part in mining operations, because they dislocate the ore-bearing "verticals," and are themselves related to extensive ore-bodies on the contact. The picture they present is that of white bands of crushed quartz, cutting through everything, as distinguished from the verticals, which are like pink ribbons of rhodochrosite, traversing the sedimentaries with difficulty because of their faulting by these cross-veins.

The latter are built upon fault-lines marking movements greater in extent than those accompanying the older, ore-bearing fractures. They are essentially quartz-veins. A variable amount of crushed country accompanies the quartz. Rhodochrosite and valuable* sulphides are notably absent. When

* Many assays were made of the pyrites from cross-veins encountered in crosscuts. Traces of gold and 4 to 8 ounces of silver were about the best results. Oc-

seen, they represent broken fragments of pay-veins traversed by the path of the cross-vein. Next to quartz, iron pyrites is their most characteristic mineral. The pyrite is in a crumbly, easily disintegrated state, very unlike the solid crystalline condition in which it appears amid the ore of the verticals. Sometimes no foreign minerals are present in notable quantity, and the cross-vein is simply a seam of crushed country, softening into mud. This the Cornishman would call "fluccan."

Fig. 17 represents a cross-vein in the breast of a drift on the Jumbo No. 3 vein, which had met the cross-vein and had been faulted by it. The Jumbo vein was intercepted by this drift 8 feet further ahead. Three breaks are noticeable, each marked by faulting. The cross-vein itself lies between two unlike beds of lime, marking a fault which evidence elsewhere along the drift showed to be 6 feet. The lime on the foot-wall is separated from a series of thin beds, exhibiting a good deal of variety, by a curving line of selvage, following a fault whose throw is 4 feet. Then come beds of shale, crystalline lime and sandstone, traversed by a dislocation of 10 inches. In each case the down-throw is on the hanging-wall. The cross-vein is almost vertical and carries fragmentary rhodochrosite torn from the Jumbo vein.

Fig. 18 illustrates another cross-vein in the act of cutting through the Jumbo vein. The line of the former is deflected in breaking through the latter. To the left the series of beds consists of blocky lime, lime shale, broken lime, limestone, black shale, closely laminated lime and light gray sandstone. On the right this series is seen to be succeeded by beds of dark sandstone. The fault along the cross-vein is 4½ feet, obliterating the displacement which must have accompanied the line of the ore-bearing vein. The latter is thrown its own width, 10 inches. It has been shattered by the cross-vein, but appears to have been repaired and reconsolidated (since its displacement) by

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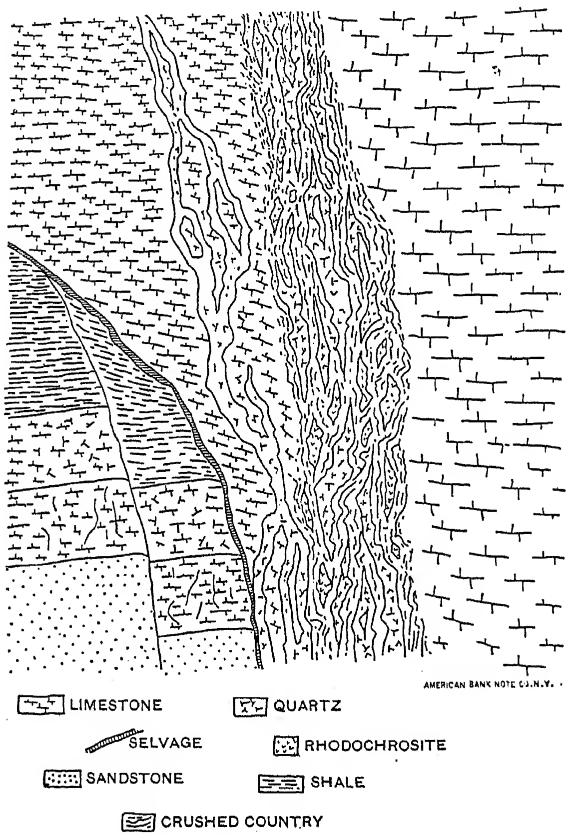
casionally good assays were ob ained, but the pieces from which these samples came were invariably marked by the presence of fragments torn from the pay-veins. Nevertheless, from a scientific st ndpoint, pyrite containing a few ounces of silver is as much ore as blende or galena, carrying much higher values. In this connection it may be added that the pyrite of the verticals is not notably silver- or gold-bearing unless admixed with copper pyrites, blende or galena. The pyrite and quartz, whether in the cross-veins or the verticals, is not a sign of valuable ore.

healing seams of quartz, which now ramify through the previously broken rhodochrosite. This cross-vein was distant 28 feet from the one described above.

The comparatively small scale of the rock-formation affords many excellent illustrations of phenomena usually requiring

Fig. 17.

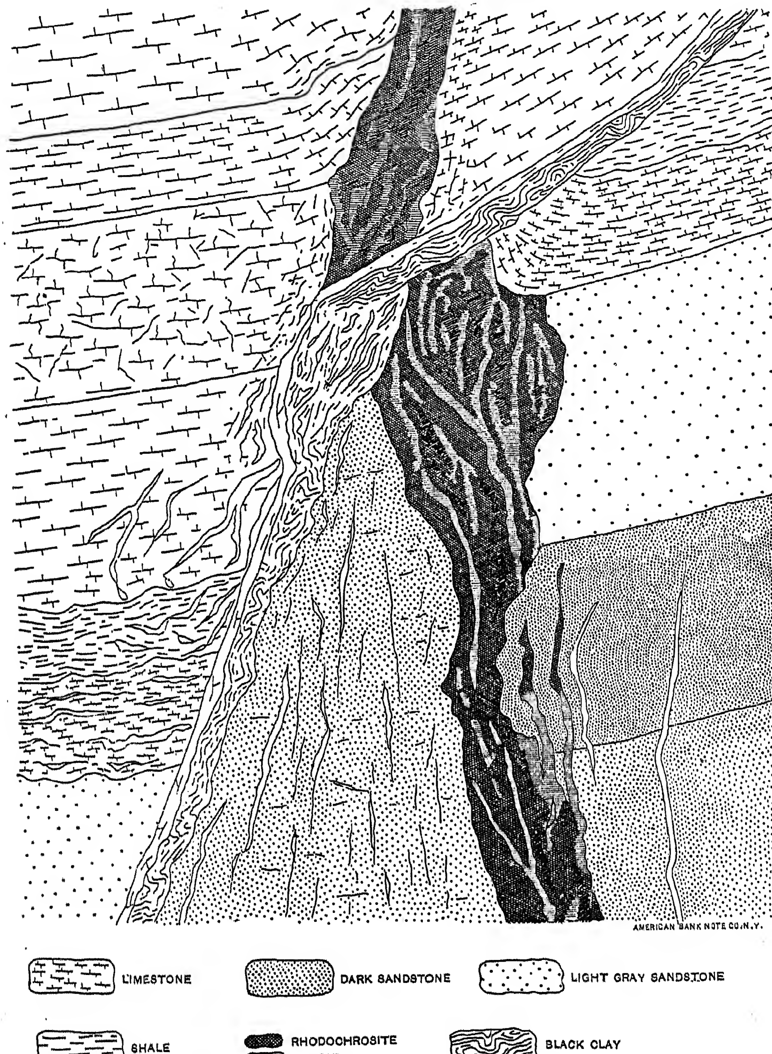


A Cross-vein in Limestone.

large areas for their exemplification. Some of the dislocations are so slight that they can be seen to die out in mere distortions. See Figs. 19 and 20. The former is particularly instructive. It shows the sides of a cross-cut which joins the workings on the Hiawatha and Enterprise veins. Near the floor of the cross-cut there is a fault of about 5 inches which follows a seam of

clay nearly 1 inch thick. This dislocation diminishes upward; the thin beds break less and bend more, until finally the verti-

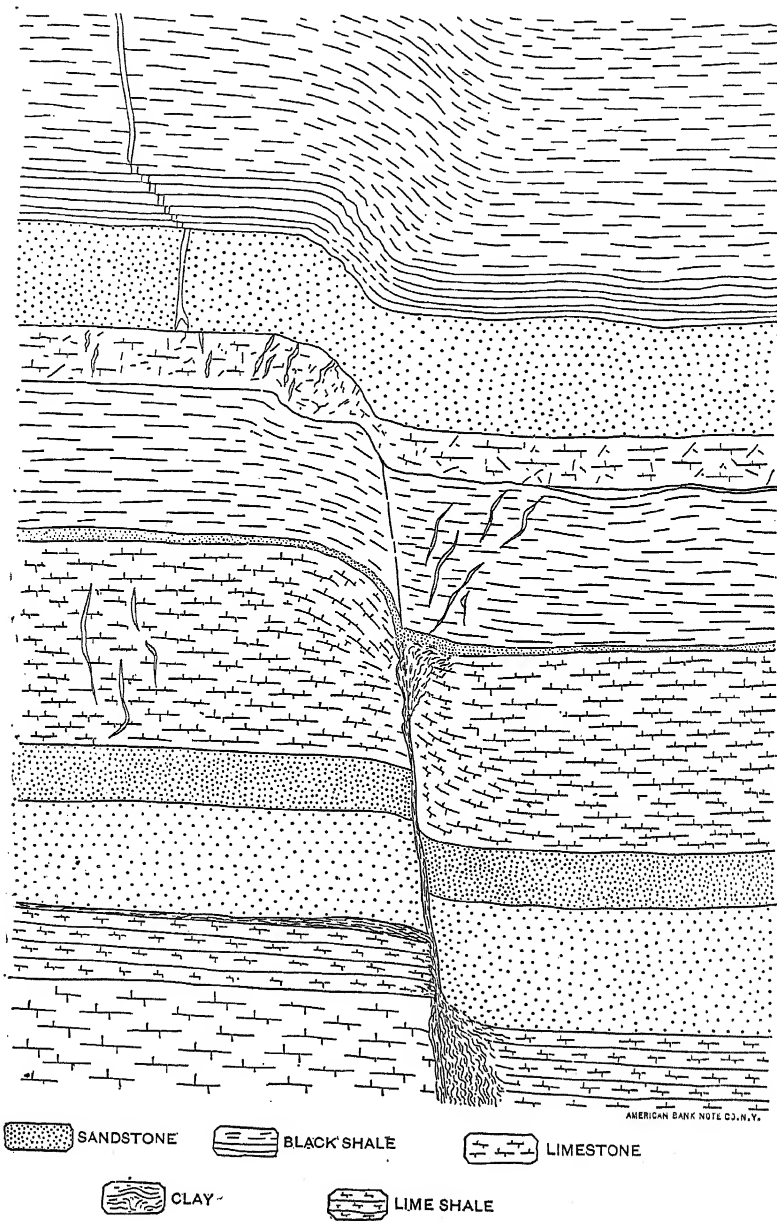
Fig. 18.



The Jumbo, No. 3, Vein Faulted by a Cross-vein.

cal displacement fades out in horizontal shifting. The latter is clearly evidenced by the minute multiple step-faulting of a

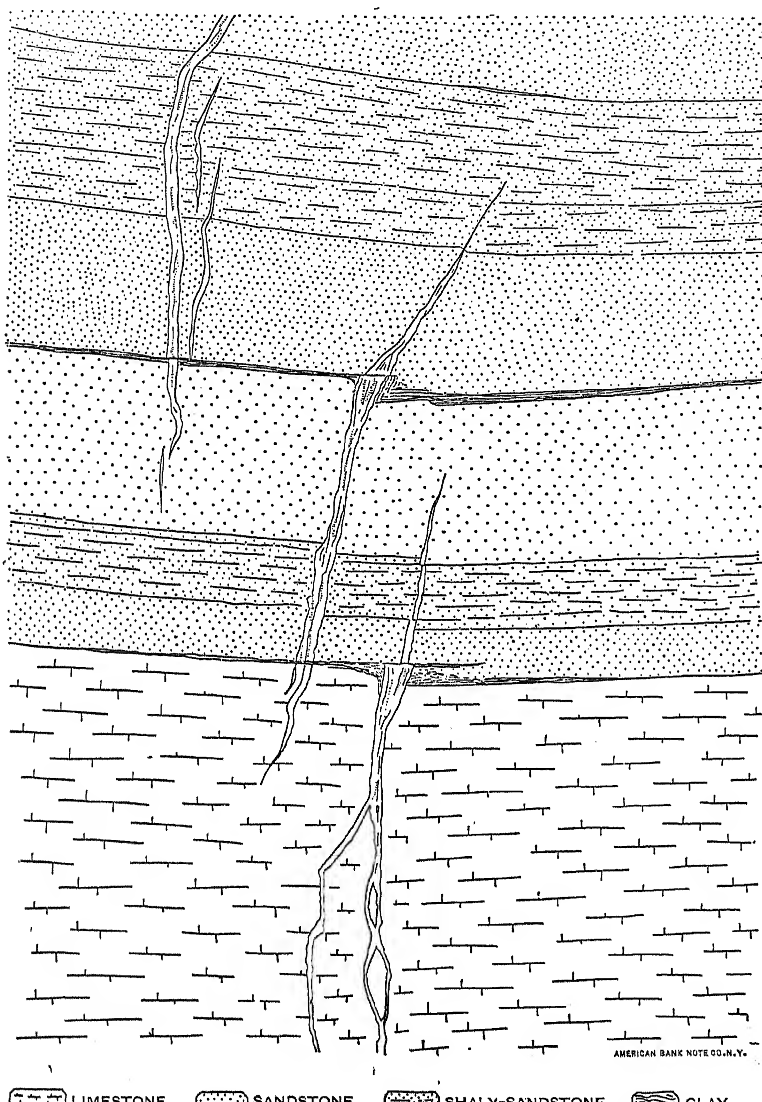
Fig. 19.



Disappearance of a Fault.

quartz-seam in the shale, near the roof of the cross-cut, which

FIG. 20.

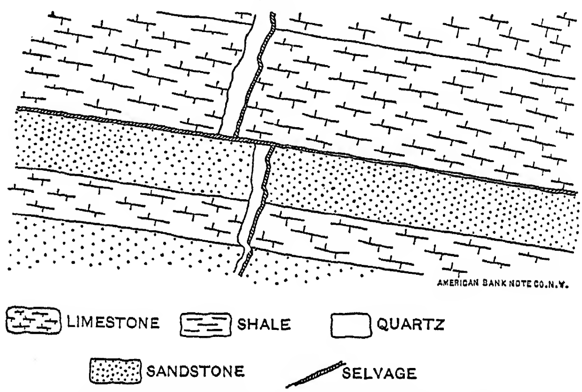


Disappearance of a Fault.

indicates that the flexibility of the shale enabled it to resist

fracturing by dissipating the force of the vertical movement in a sliding of the laminæ over one another. Fig. 20 conveys the same lesson. The three quartz-seams here shown follow lines of lessening vertical movement; the distinct faulting of the beds in the lower part of the section dwindling until in the upper portion only a slight distortion of the country is to be discerned. These examples illustrate how large faults may gradually disappear, and suggests why the vein-systems of Newman hill terminate in their approach to the horizon known as the "contact."

FIG. 21.



Faulting of Cross-vein along Bedding-plane.

The cross-veins are themselves displaced in some instances by slight later movements which have taken place along the bedding-planes of the country, such as indicated in Fig. 21, where a small south-dipping cross-vein is thrown 5 inches along the parting between beds of limestone and sandstone.

The dislocation of the ore-bearing veins by the cross-veins necessitates the employment of scientific methods in mining. The mine-workings of Newman hill are needlessly tortuous and complex, because most of them have been directed by men blind to the indications of geological structure. The expensive results of a bewilderment due to this cause are well illustrated in the case of the Hiawatha vein, between raises No. 9 and No. 11 on the main level. This particular instance is quoted because it appears in Mr. J. B. Farish's paper on Newman hill.* His

* *Proceedings of the Colorado Scientific Society*, vol. iv., p. 159.

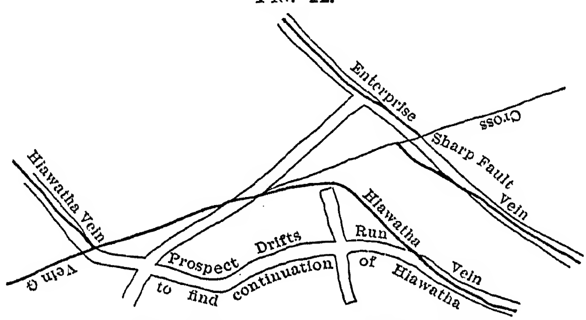
drawing is here represented (in Fig. 22), together with another (Fig. 23) based on a very close sifting of the evidence, accompanied by a careful survey.

Referring to Fig 22, I quote Mr. Farish's description:

"It illustrates the occurrence of a fault in the Enterprise vein and the deflection of the Hiawatha vein by the same cross-fissure. The break in the Enterprise vein is seen to be sharp, while the Hiawatha vein is not faulted, but makes along the cross-vein for nearly 100 feet before emerging from its walls and resuming the original course."

The vein called the Enterprise in the above description is not the Enterprise, but the Songbird. The former vein has

Fig. 22.



Mr. Farish's Drawing of the Hiawatha Vein.

never been found in this particular part of the mine, because it breaks up into unimportant stringers and is not definitely recognized until nearly 400 feet further north. The Enterprise dips east slightly, the Songbird dips west flatly. In their strike northward the two veins unite.

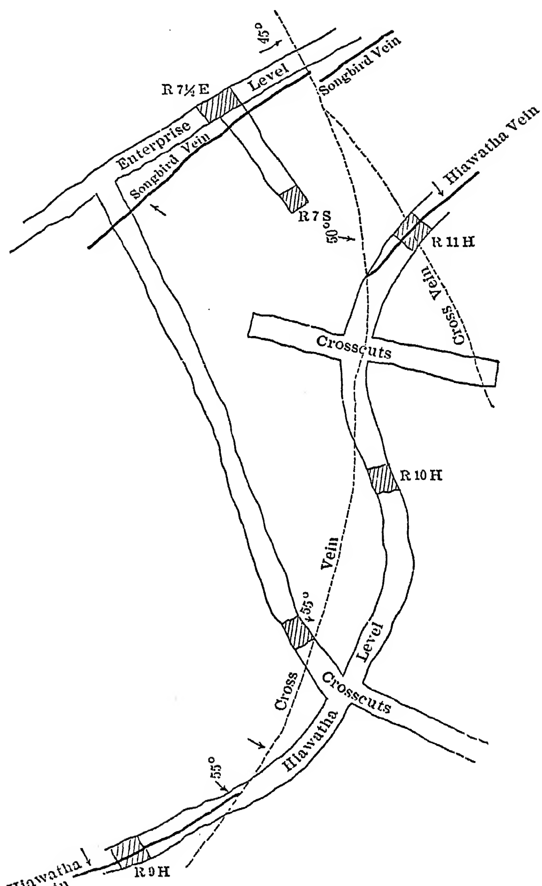
The cross-vein faults both. If Fig. 22 were correct the formation of the Hiawatha would be later than that of the cross-vein, and the latter must be later than that of the so-called Enterprise vein, and the ore-bearing veins would not be of contemporaneous origin. Such, however, are not the facts.

The cross-vein strikes the Songbird almost at right angles and throws it to the right. When it meets the Hiawatha, it throws that vein to the left. The Hiawatha and Songbird dip in opposite directions. Both faultings are in accord with Carnall and Schmidt's rule.*

* In *Fault-Rules*, by Francis T. Freeland, *Trans.*, xxi., 499.

The representation (in Fig. 22) of the Hiawatha as following the cross-vein is, in my judgment, incorrect; the error having been made by mistaking the "drag" for ore in place. The same error is repeated in another drawing (Fig. 24) by Mr.

FIG. 23.



The Faulting of the Hiawatha and Songbird Veins by a Cross-vein.

Farish,* which Professor Kemp has perpetuated in his treatise on ore-deposits.† The value of the latter as a book of reference makes it imperative that the mistake should be pointed out.

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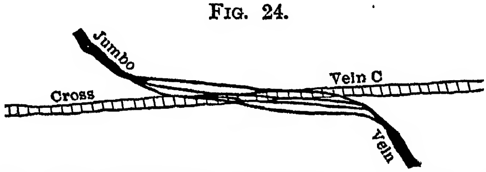
* Pages 158 and 159 of the paper cited.

† *The Ore-Deposits of the United States*, by James F. Kemp, p. 20.

Mr. Farish says that the occurrence represented by him in Fig. 24 is an instance of the "bend of a vertical pay-vein as it approaches an intersecting cross-vein," and also, that "the Jumbo vein, as it approaches and departs from the cross-fissure is considerably disintegrated, the numerous seams and stringers striking diagonally through the cross-veins." Professor Kemp uses this drawing to exemplify the faulting of one vein by another. Mr. Farish's description would make it appear that the cross-vein existed prior to the Jumbo vein, and that the latter was bent by the former, which it traversed in seams and stringers. Yet the drawing, even as it is, represents no fault, but merely the bending of one vein by another, of possibly contemporaneous origin.

The real conditions are represented in Fig. 26, which shows

FIG. 24.



Mr. Farish's Drawing of the Jumbo and a Cross-vein.

how the cross-vein faulting the Jumbo carries shattered fragments of the latter between its two dislocated parts. The confusion caused by Mr. Farish's attempt to distinguish between cross-veins that fault pay-veins, of necessarily older origin, and cross-veins followed by pay-veins, of consequently later formation, is due, I think, to a failure to recognize a very simple feature of vein-faulting. I refer to those fragments of the ore of the older vein which have been broken off by the fault-fissure and are found scattered amid the newer filling of the latter along that part of its course which lies between the two disrupted portions of the older vein. This is the "drag," which is so valuable an aid in mining, because it enables the miner to trace the direction of the throw.

The faulting of the "verticals" by cross-veins is a prominent feature in the Enterprise mine. When half a dozen drifts were running on the several pay-veins a fault was encountered about once per month. In other words, the distance between the cross-veins averaged from 65 to 100 feet. Failure to apply the

elementary rules of faulting and to make such observations, measurements and calculations as would serve to identify a cross-vein as it passed through the series of nearly parallel verticals, has caused much unnecessary expenditure on Newman hill. More broken ends of faulted-veins were happened upon by accident in "drifting" than were discovered by intelligent search. The practical importance in mining of the study and interpretation of geological structure is forcibly emphasized by this record of experience.

Out of over two hundred instances of faulting noted during my twelve months' direction of the Enterprise mine, I detected only one (and that an uncertain) exception to Carnall and Schmidt's rule, which is stated by Mr. Freeland as follows:*

> "If the fault be encountered on its hanging-wall side after breaking through it, prospect toward the hanging-wall side of the vein; on the contrary, if from the foot-wall side, then prospect toward the foot-wall side of the vein."

"This rule," as Mr. Freeland says, "applies only to normal faults, and is, in addition, subject to an important exception."† I do not cite it as the statement of a universal law. In other districts other rules may be found applicable. Nor is it necessarily to be expected that in any one district a single rule only will obtain. But if in a given locality the prevalence of a given fault-rule (expressing the habit of the veins in that locality) can be demonstrated, a working-hypothesis of immense value is thereby furnished to the miner. My remarks in this connection are to be taken, therefore, as applicable only to the district here under consideration, or to other districts in which similar conditions may be determined by experience.

When a cross-vein intersects a pay-vein at an acute angle it is practicable to follow the trail of the latter, as evidenced by the scattered fragments constituting the "drag." When, however, the faulting occurs at a large angle there is less evidence of the direction of the throw, and a level driven on the cross-

* *Trans.*, xxi., 499.

† "Normal faults" are those in which the rock forming the hanging-wall of the fault has moved downward along the foot-wall of the fault. The exception to Carnall and Schmidt's rule mentioned by Mr. Freeland in the above quotation is that of an obtuse fault-angle. The rule might be so stated that this would be no exception, as the case is simply one of an acute-angled fault, *approached from the other side.*

vein until it meets the other portion of the disrupted vertical would involve turns too sharp for subsequent use in tramming. It might thus be necessary, after finding the continuation of the pay-vein, to run a new piece of level for practical mining purposes. But if the direction of the throw be known from pre-

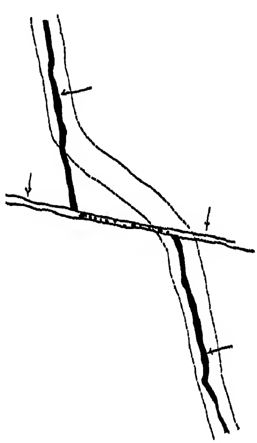
Fig. 26.

Faulting of the Jumbo by a Cross-vein.

vious experience, the heading can be so turned as to strike the vein beyond its dislocation by a course of minimum deflection, making a practicable mine-level.

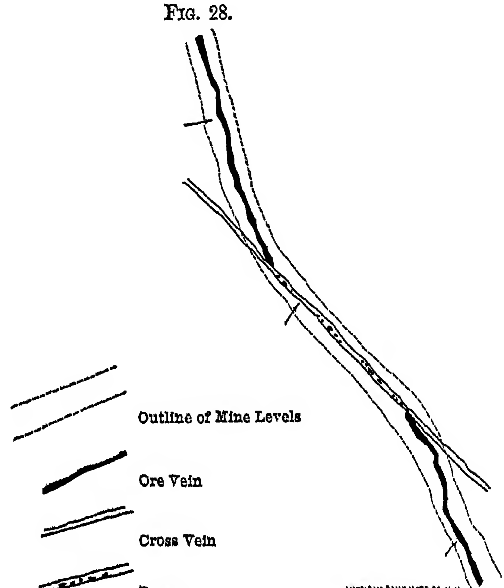
When a cross-vein meets a vertical at a very small angle a

Fig. 27.



A Vein Faulted almost at Right Angles.

Fig. 28.

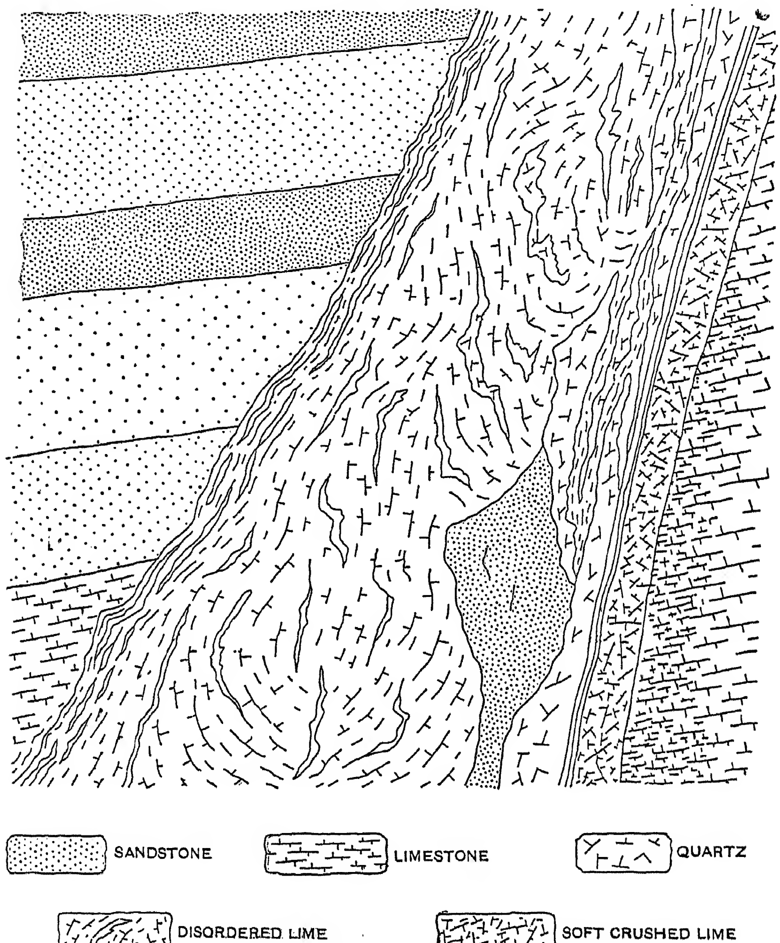


A Vein Faulted at a Small Angle.

slight throw, as measured at right angles to the strike, becomes sufficient to produce a wide separation between the broken ends as measured along the course of the pay-vein. In such cases it is easy for the unobservant miner to be misled by the

unapparent deviation in his level and to mistake a cross-vein carrying fragments of lode-matter for the pay-vein itself. (See the diagrammatic sketches in Figs. 27 and 28.)

FIG. 29.

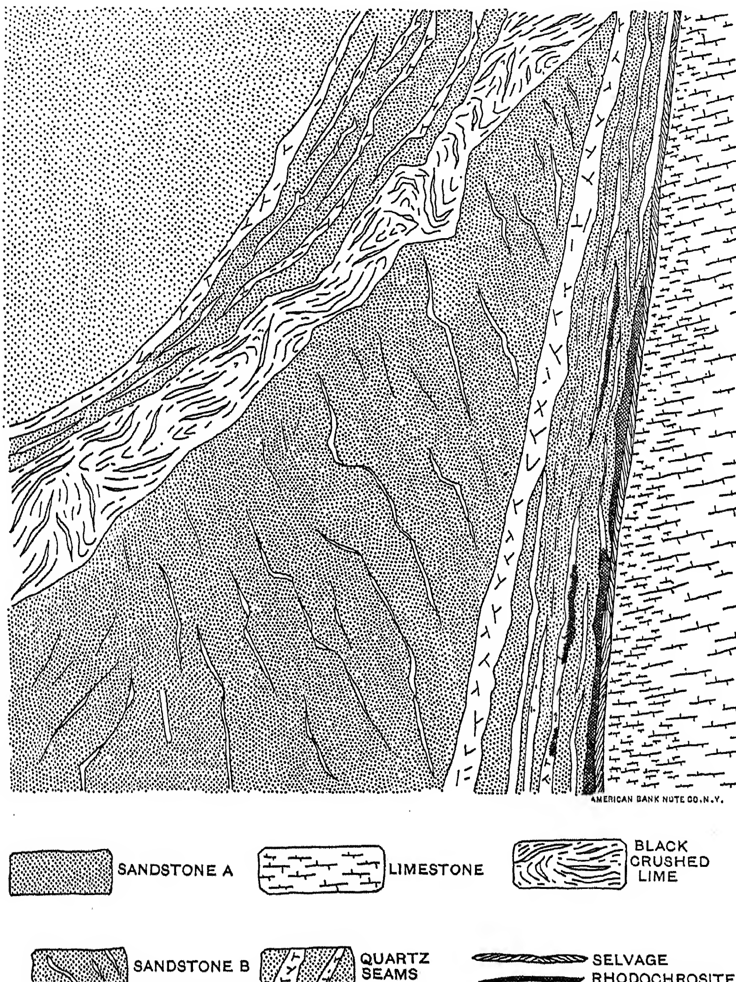


Cross-vein as seen in the Breast of a Drift on Jumbo No. 2.

The rhodochrosite is a great aid in tracing the ore-bearing veins because its bright color renders it readily distinguishable amid the white quartz or the dark shattered country of the cross-veins, and its absence distinguishes the latter just as its presence characterizes the pay-veins.

Figs. 29, 30 and 31 illustrate the behavior of two cross-veins, encountered in following the Jumbo No. 2 lode. Fig. 29 shows the cross-vein in the breast of the level. It consists of 2 to 3

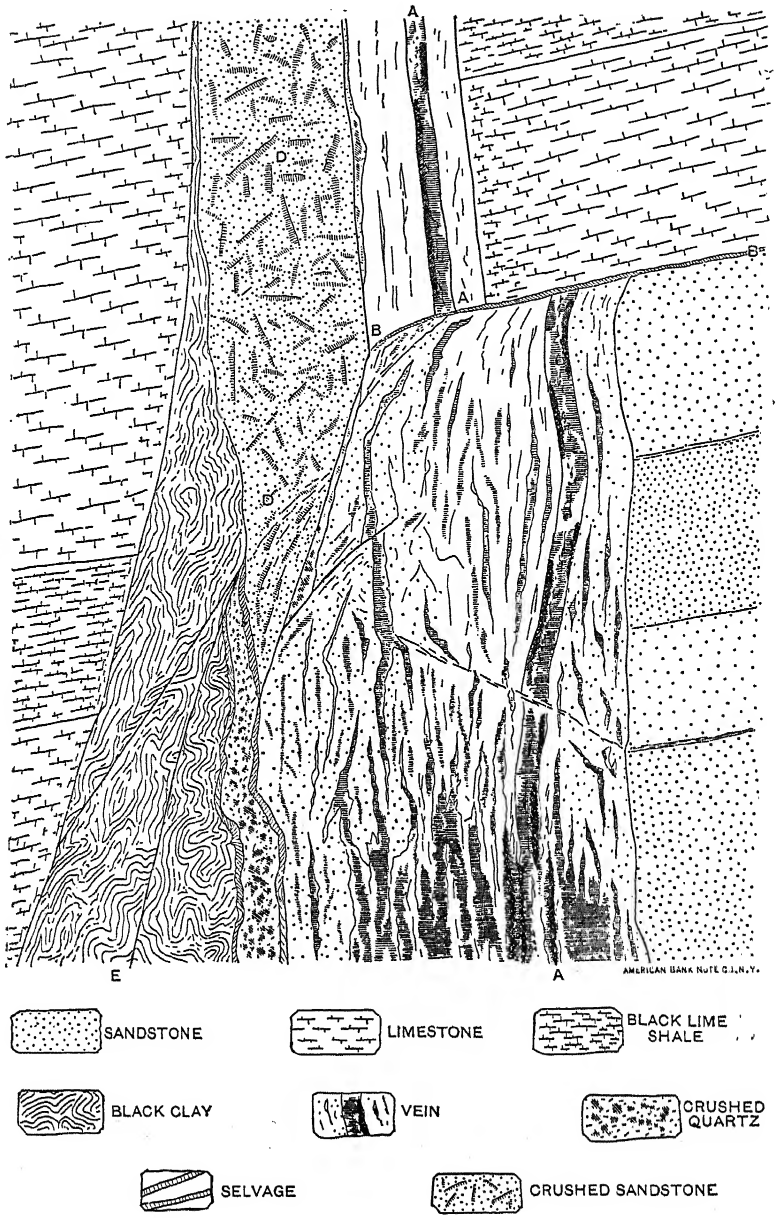
Fig. 30.



Two Cross-veins Approaching Each Other.

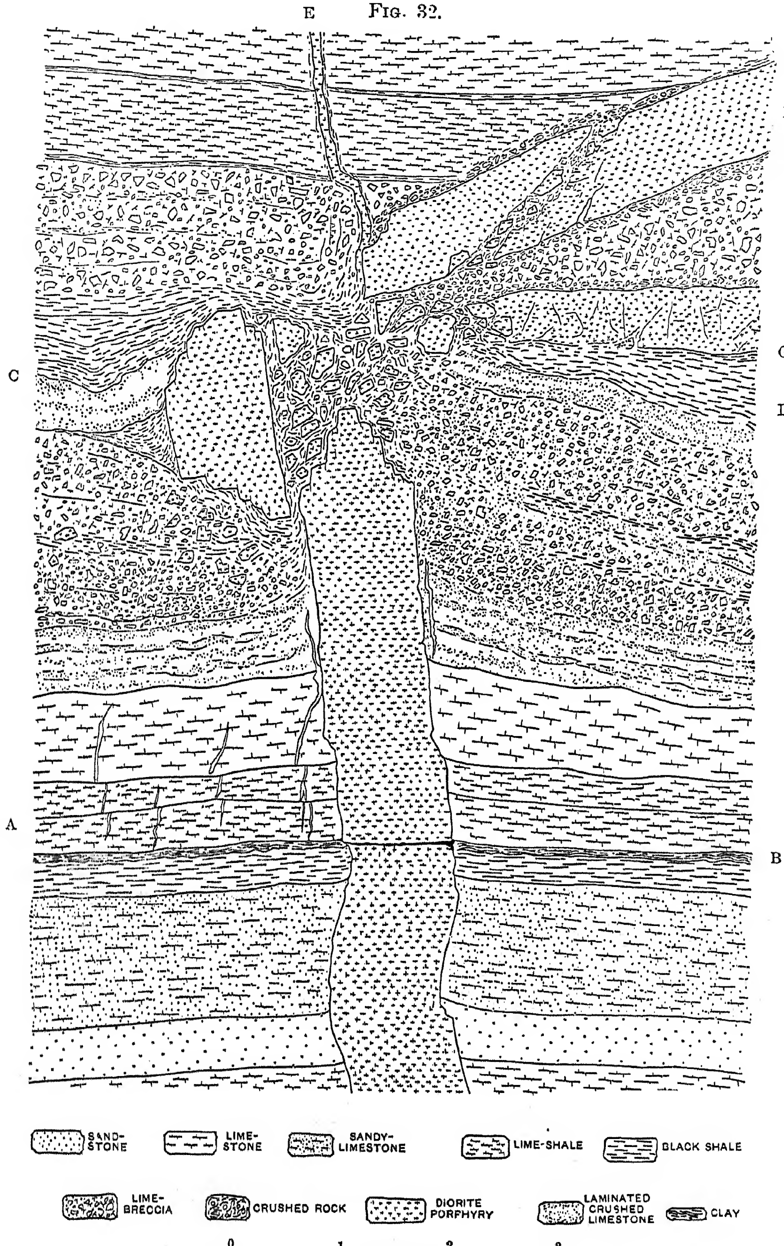
feet of crushed country, enclosing stringers of quartz. There is a fault, but its extent cannot be determined. The wedge of sandstone observable near the floor of the drift marks the be-

FIG. 31.



A Vertical and a Cross-vein Intersecting.

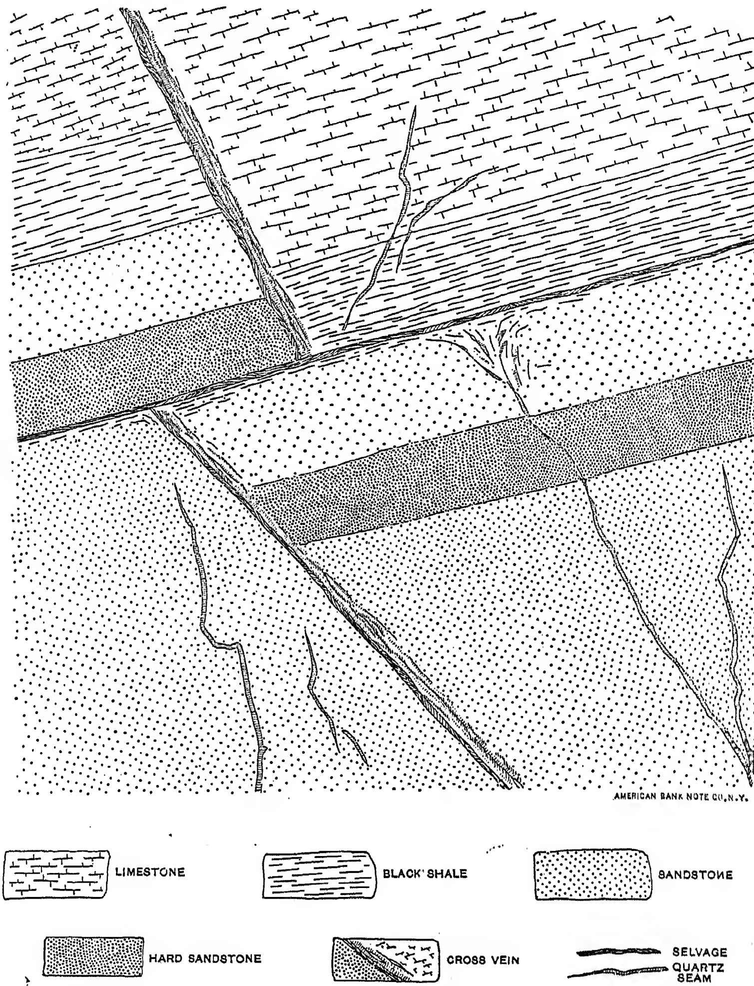
Fig. 32.



Porphyrite Dike at the Contact.

ginnings of a separation which Fig. 30 shows to be the departure of one of two consolidated cross-veins. One of these

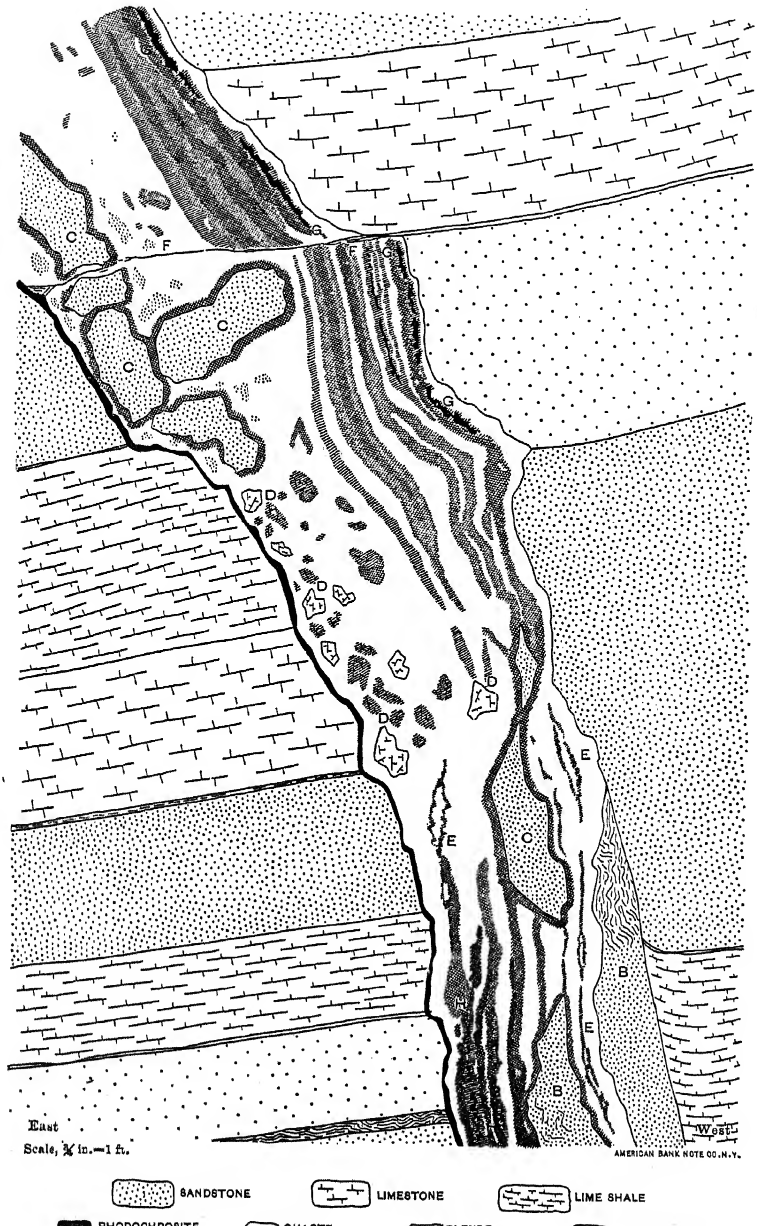
FIG. 33.



BREAST OF CROSSCUT WEST OF ENTERPRISE LEVEL, APRIL 10TH, 1894

began at this time to exhibit pieces of rhodochrosite, which soon afterwards led to the finding of the vein. Fig. 31 illustrates another cross-vein. The structure is complicated. The narrow seam, A A, and its enclosing vein-filling have been

Fig. 34.



KITCHEN VEIN

faulted along a bedding plane, B B, and subsequently a movement along the hanging-wall of the cross-vein has led to another dislocation, a down-throw of the country and a distortion of the former line of movement. It is interesting to note that the secondary companion cross-vein thus formed consists of crushed sandstone, D, in the upper part of the section, and of black lime, E, which has been crushed into clay, in the lower part. This change corresponds with the alternation of rock in the enclosing country.

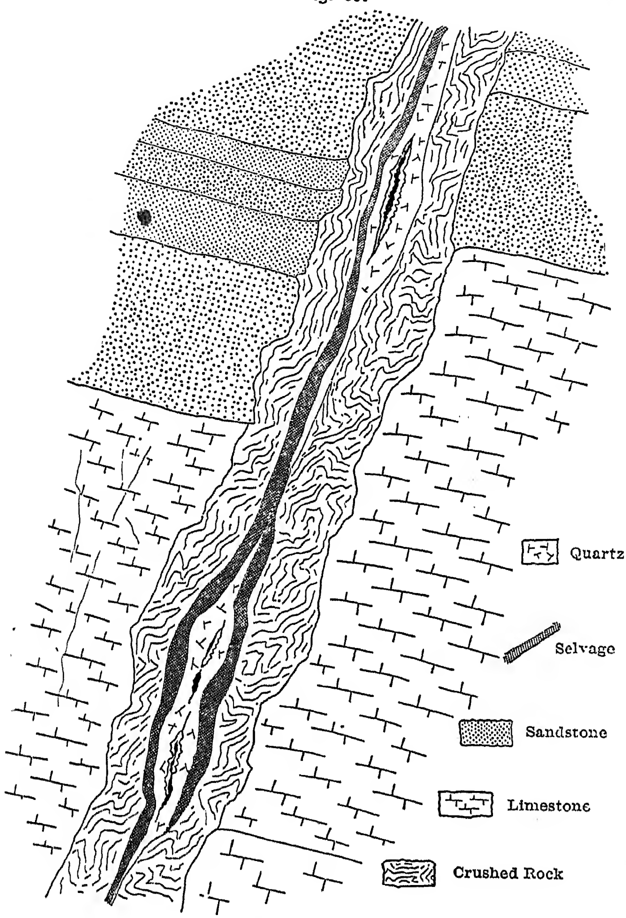
Fig. 33 illustrates a fine example of double faulting. The steep cross-vein has caused a displacement of a little over 2 feet, which is very clearly marked by the dark band of sandstone. Subsequently a shifting along bedding-plane has faulted the cross-vein about 20 inches.

Fig. 34 is a section of the Kitchen-vein, as seen in the face of a south drift. This vein dips westward. The example is of value because of the inclusion of fragments of country-rock. Of these, some (CC) are sandstone and others (DD) lime. Their distribution corresponds with that of the rocks on the foot-wall, and suggests that they were torn off the enclosing country. The mineralization of the rock and the substitution of ore is suggested by the fact that each of these included pieces has a well-marked rim of rhodochrosite. On the hanging-wall there is no parting from the country; on the foot-wall the ore is readily detached, because of the existence of a continuous selvage of black mud. Ribboning, due to alternations of quartz, rhodochrosite and blende, is well marked in the upper hanging-wall portion of the section. This vein follows a fault-fracture, the throw of which exceeds 5 feet, an amount unusual to the pay-veins of the Enterprise mine.

The occurrence of dikes and tongues of porphyrite has been noted incidentally. The deep shafts and bore-holes have encountered large thicknesses of it among the sedimentary rocks underlying the horizon of the existing mine-workings. The sedimentaries have been found hardened and otherwise affected in the vicinity of the eruptive rock, so that sandstone appears as quartzite and limestone is converted into marble. What the real character of this body of porphyrite may be cannot be determined from the evidence at my disposal. The more general observations of the gentlemen of the Geological Survey will

doubtless throw much light on the matter. To the mining engineer one fact stands out very prominently, that whether in their strike northward, approaching the porphyrite of Silver

Fig. 35.



A Typical Crossvein.

creek, or in their extension downward, toward the lower-lying body of that rock, the veins become impoverished of pay-ore. In view of the general recognition of the beneficent association of veins with eruptive rocks, this is very instructive. The explanation suggests itself that though the shattering of the country accompanying the intrusion of the porphyrite aided

vein-formation, because of the passageways thereby developed for the circulation of underground waters, yet at the same time the immediate neighborhood of hot masses of eruptive material did not favor the precipitation of ore, and that the best deposition took place at such a distance as permitted the cooling of the waters and the consequent laying down of the metals they carried in solution.

The veins penetrate the porphyry. The contact in many places carries fragments of porphyry. These facts prove, as would be readily surmised, that vein-formation succeeded the intrusion of the porphyry.

In the stopes irregular tongues of porphyrite were occasionally observable. As they only followed the vein for a short distance in its strike and dip, it was not possible to find out much about them. Such occurrences are seen in Figs. 5, 15 and 16. Fig. 32 is a drawing, which I made with particular care, showing how a dike is shattered at the contact. The porphyrite is 14 inches wide. Along the parting AB it is fractured, and a clay selvage across it suggests later movement (in the direction of the strike, so that the cross-section does not show it). At the level of CD the dike is smashed to pieces. A disturbance along the contact has shifted some of the fragments. The regular passage of the dike upward has been interfered with at the contact, and, previous to the shattering, it seems to have divided into branches, of which three (terminating in the sketch at E, F and G) still survive. The contact itself, here marked by a thickness of nearly 5 feet of crushed shale, crushed sandstone and lime, contains numerous fragments of porphyrite. Apart from its bearing on the origin of the dikes, this drawing illustrates how the contact barred the progress upward of fissuring of every kind.

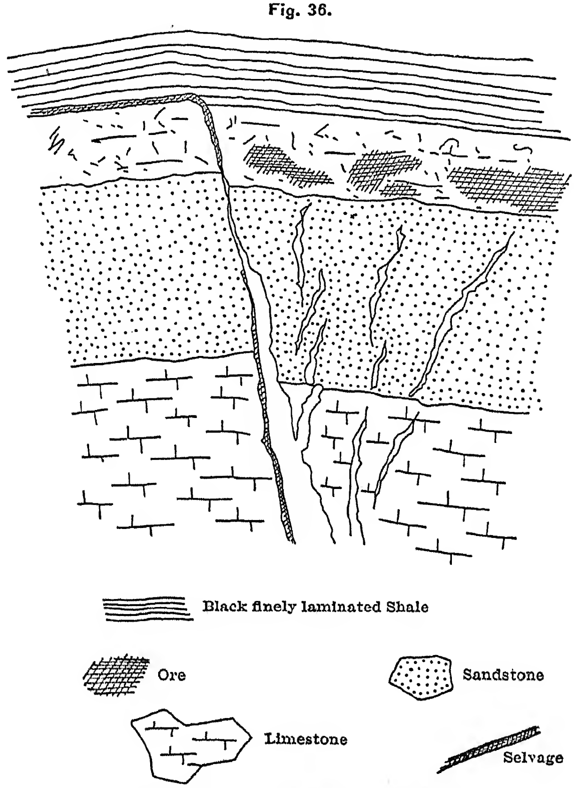
VI.—The Contact.

As the ore-bearing veins are followed by the mine-workings* they are found to split up, weaken and become impoverished

* Entrance to the mines is made through long adits which aim to cut the veins at a depth of 50 to 100 feet below the contact. Hence the regular vein-structure is first seen, and the other complications come under notice as the stopes and raises are ascended. This order has been observed in describing them here.

at a certain horizon, lying from 5 to 20 feet below a zone of crushed rock, in which are enclosed ore-bodies richer than those of the veins themselves. It is rarely that even diminished seams of ore survive so as to connect with the ore on this contact. Ordinarily the highest stope on the "vertical" is separated from the nearest stope on the "contact" by a few feet of black shale

Fig. 36.



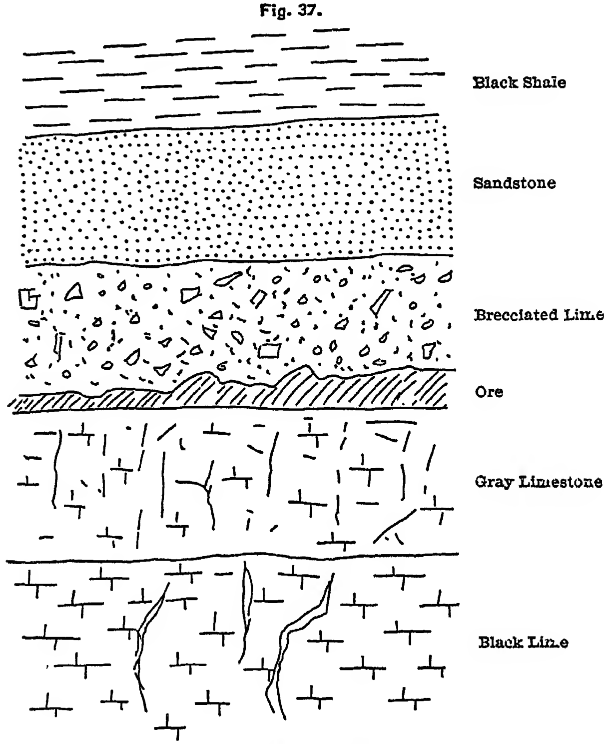
JUMBO NO. 2. VEIN AT THE CONTACT

and lime, containing, at best, only scattered remnants of the vein. Figs. 36 and 37 give simple examples. Occasional sections show no connection between the vertical and the contact overhead. Such small quartz or rhodochrosite threads as do occur have no continuity.

The cross-veins behave in relation to this contact in a manner very similar to the pay-veins. Since they are barren of ore, there are no workings which follow the cross-veins in their

immediate approach to the contact, but there is scattered evidence* sufficient to confirm this statement. There is, however, a notable difference between the behavior of the two systems of vein fissuring when they reach the contact. The economically important observation has been made that the cross-veins are less regularly topped by ore-bodies along the

Fig. 37.



A TYPICAL CONTACT

contact; but, on the other hand, such ore-bodies as have been found over cross-veins have been usually larger and certainly richer than those apparently related to underlying verticals.

The dislocating influence of the pay-veins appears to die out in its approach to the contact-zone, so that the latter undergoes a mere undulation or roll along its bedding-planes. At the same time the pay-veins become unrecognizable, even as minute

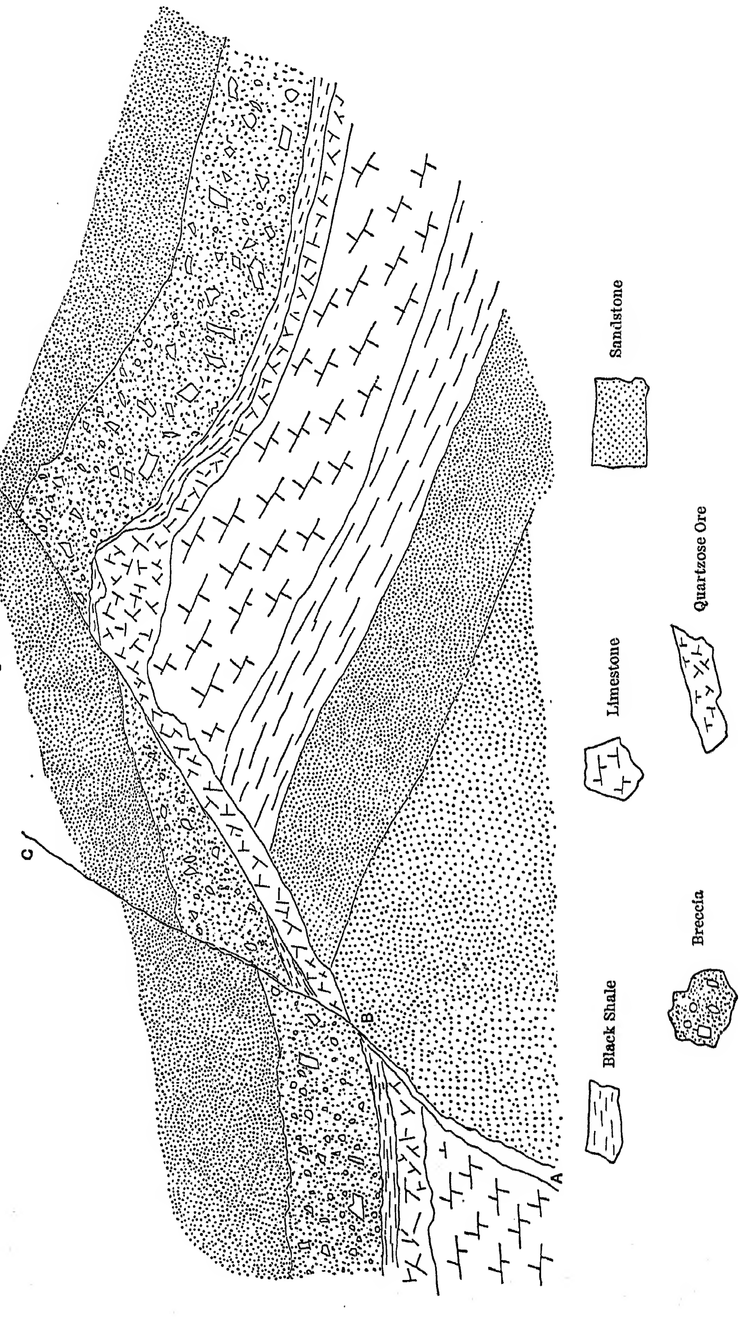
* Obtained from stopes under the contact, in places where a cross-vein happens to cut a vertical as it is dying out.

seams or partings, in the rocks above the contact. In the case of the cross-veins, however, a more serious disturbance of the contact-zone is indicated by violent bends and occasional step-faults. The cross-vein itself, though it does not reappear above the contact as a strong quartz-seam, is yet represented by a well-developed division in the country accompanied by selvage. Fig. 38 represents a section along the contact, where it is dislocated by a cross-vein. The latter, A B, splits at the contact, both branches, B C and B D, following fault-lines. The ore of the contact, which is here related to the Jumbo No. 2 vein, consists of low-grade quartz from a couple of inches to more than a foot in thickness, overlain by a thin bed of black shale and underlain by limestone. Above the shale is a bed of breccia, about 2 feet thick, composed of fragments of limestone, with occasional pieces of porphyrite and shale. It will be noticed how the quartz-ore follows the bedding and the connecting fault-fissures.

When the contact was first penetrated by the shafts sunk from the surface (through the drift covering the sandstones and limestones which contain the ore-deposits) it was supposed to be a "flat vein." When later developments proved it to conform to the bedding of the country, it became recognized as distinct from the other vein-systems which cut the bedding at a right angle. Hence grew the idea that there existed here a limestone-shale contact similar to that of Aspen and resembling the ore-bearing zones in the Carboniferous blue lime of Leadville. More extensive exploration dissipated the hope of a continuous ore-bed, but although the occurrence of narrow channels of rich mineral, ramifying through the brecciated beds of the contact, was demonstrated, it was not at first seen how clearly these corresponded with the strike of the two vein-systems, the upward course of which was terminated by the contact. The recognition of this relationship was a key unravelling many perplexities, and a light to the intelligent exploration of a territory of complex geological structure.

The contact is not, as the term might imply, a continuous plane of division between two rock formations, nor does it mark the parting between the two adjoining beds of ore formation. The accounts which have been given of a "contact-limestone," overlain by a "drab shale" and underlain by a "finely lamina-

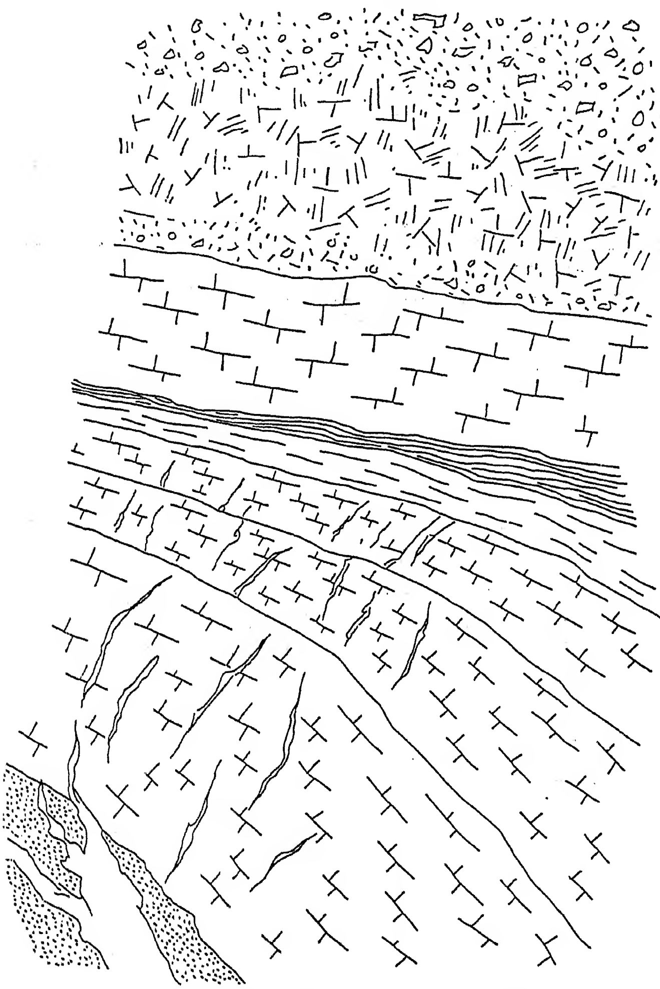
Fig. 38.



A Crossvein at the Contact.

ted shale,"* may describe certain sections of this ore-bearing horizon, but they do not characterize it as a whole, and they

Fig. 39.



THE CONTACT ABOVE THE ENTERPRISE VEIN

give a misleading idea of its real nature. In the three sections already given, in Figs. 36, 37 and 38, the contact is found re-

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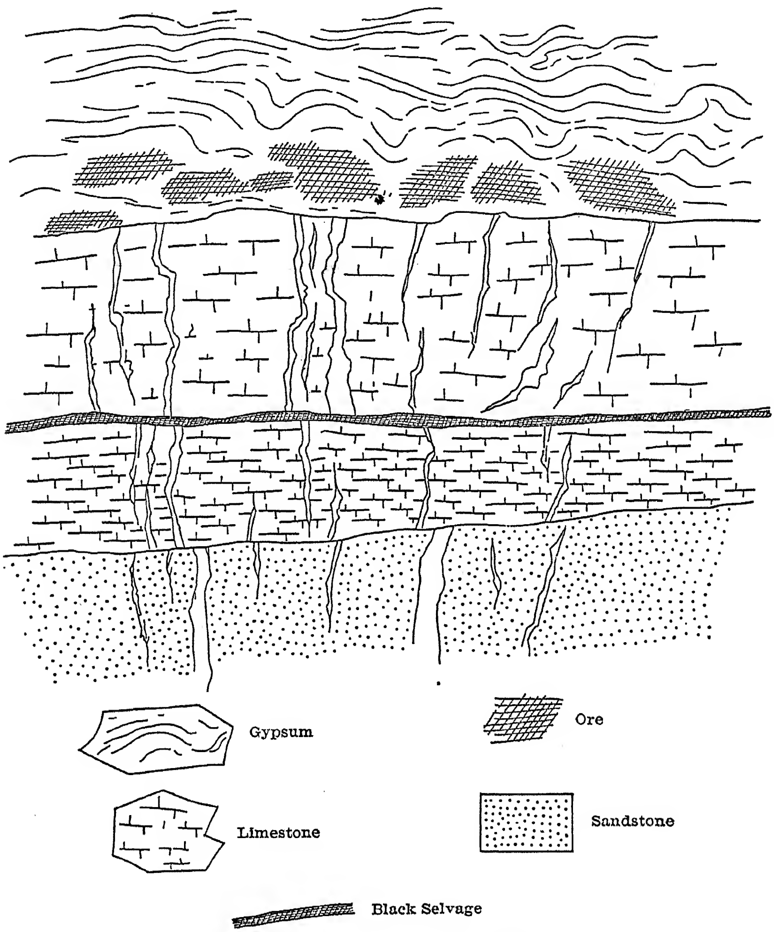
* J. B. Farish, "On the Ore-Deposits of Newman Hill." *Proceedings Colorado Scientific Society*, vol. iv., and James F. Kemp, *Ore-Deposits of the United States.*

spectively in a crystalline lime, overlain by black shale and underlain by sandstone; in a lime-breccia, overlain by sandstone and underlain by a gray limestone; and in the third instance

Fig. 40.



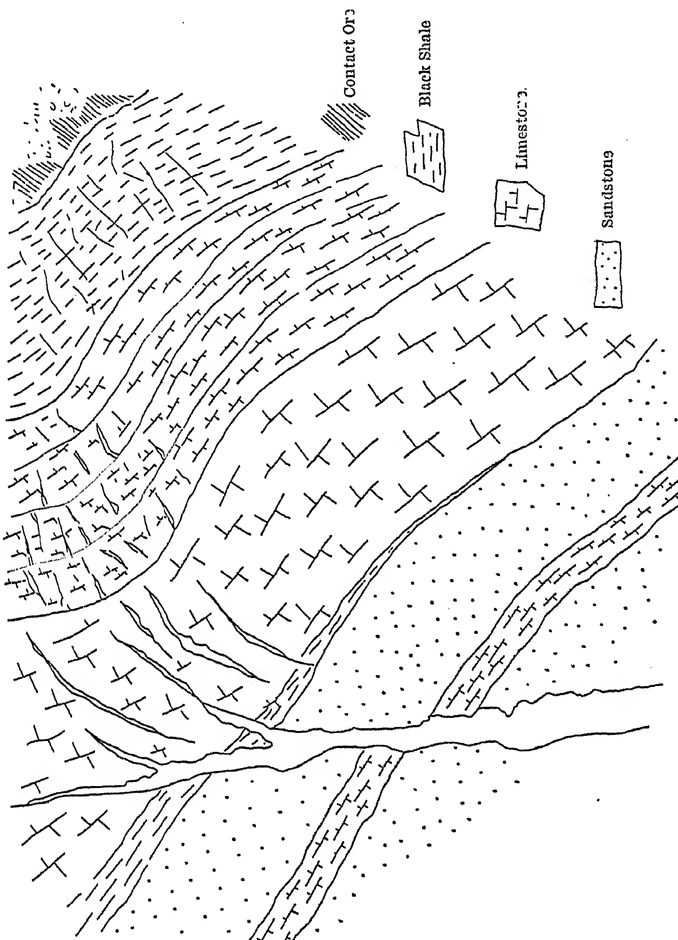
FACE OF A STOPE ON THE CONTACT

in a mass of crushed quartzose lime, covered by black shale and overlying a blocky limestone. Additional sections are now given in Figs. 39, 40 and 41. In the first of these we see the stringers thrown out by the Enterprise vein as it nears the contact, which in this case consists of brecciated lime, enclosing a low-grade quartz ore. In Fig. 41 another similar example is

given. In Fig. 40 the contact-ore lies at the base of a bed of gypsum, which in turn overlies limestone, penetrated by stringers, which come from the Jumbo No. 3 vein below.

The ore of the contact cannot be said to be confined to any

Fig. 41.

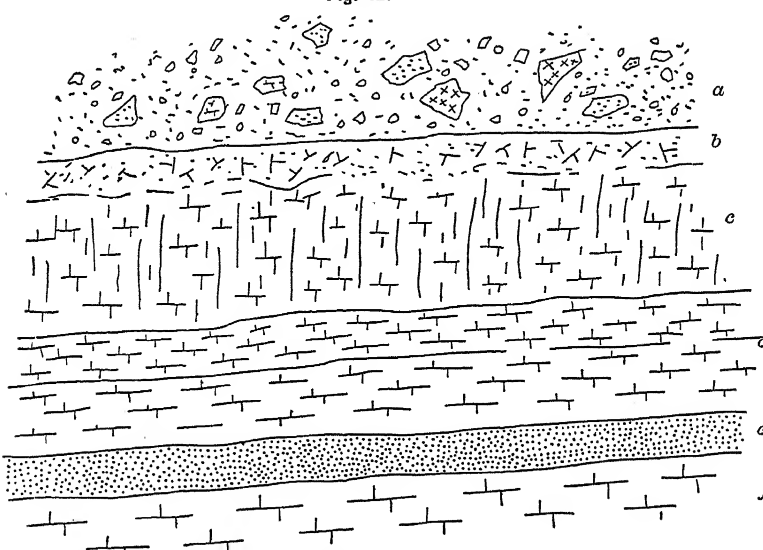


THE ENTERPRISE VEIN AT THE CONTACT.

particular encasement; but one may venture the generalization that it is to be sought for in a layer of crushed rock, which occurs along a certain horizon marked by a thinly-bedded series of black limestones and shales. The parts of the contact explored during my period of management were very frequently characterized by a distinct breccia made up principally, but not

solely, of lime-fragments. Pieces of shale and sandstone were recognizable as derived from adjacent beds; and fragments of porphyrite were traceable to neighboring intrusions of that rock. The contact above the Jumbo No. 2 frequently consisted of compact, pulverulent lime, graduating into breccia overhead and underlain by blocky lime; that above the Enterprise was often breccia, shading off into blocky lime overhead, and underlain

Fig. 42.



a, Breccia; *b*, Crushed Lime and Quartz; *c*, Lime; *d*, Laminated Lime; *e*, Sandstone; *f*, Blocky Lime.

A Barren Contact.

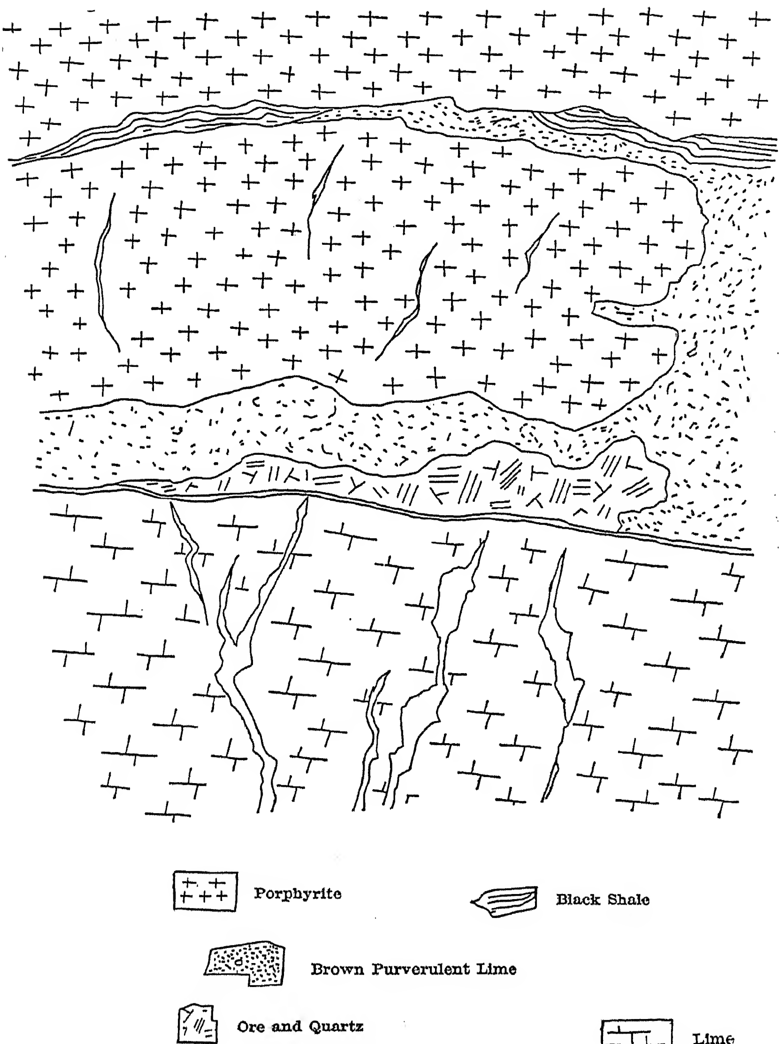
by black shale; while the ore of the Jumbo No. 3 contact was found between a powdery brown lime and a thin bed of black shale. The variability of the stratigraphical position of the ore, thus emphasized, is due to the non-persistence of individual beds.

Except when it tops the veins, the contact is barren. Fig. 42 illustrates the face of a cross-drift on the contact where no veins have enriched it. In Fig. 43 an intrusion of porphyrite is shown. The pulverulent lime indicates crushing, probably accompanying the invasion of the porphyrite.

The crystalline limestone of many sections is doubtless breccia consolidated by pressure and cemented by the underground

waters which have lined the cavities with crystals of calcite. The gypsum bed to be seen in the southwestern part of the

Fig. 43.



SECTION ABOVE THE JUMBO No. 3.

mine affords a parallel instance. A section is given in Fig. 40. The vein whose shattered termination is to be seen below the ore of the contact is the Jumbo No. 3. The gypsum thins

out and is a local occurrence having a maximum thickness of 15 feet. Its wavy, compact texture suggests an origin by a sulphatization of lime-breccia through the agency of solutions coming from neighboring ore-bearing measures.*

The foregoing general description will assist the reader's understanding of Fig. C, which represents a part of the workings on the main (also called the Enterprise) level, 98 feet above the Group tunnel. It covers an area about 1800 feet long by 350 feet wide, and was chosen for especial examination and study because the developments are more complete than in any other part of Newman hill, while at the same time they present features sufficiently typical of the ore-occurrence over the whole territory covered by the mine maps.

The levels are seen to follow three veins, the Enterprise, Songbird and Hiawatha, the dip of which is indicated by the arrows. Commencing at the crosscut from the Enterprise shaft the Enterprise vein is followed without difficulty as far as Raise 6,† where the vein dies out in weak stringers. Up to this point the stopes extend to the contact; but to the northward, stoping ceases on this vein. The vein which the level followed further on, from R8S, was once considered the Enterprise because of imperfect observation of the facts. So also the vein followed by the level further north still, from R15E to R18E, is called the Enterprise. It is a branch vein, a subordinate member of the series of many disclosed in the mine.

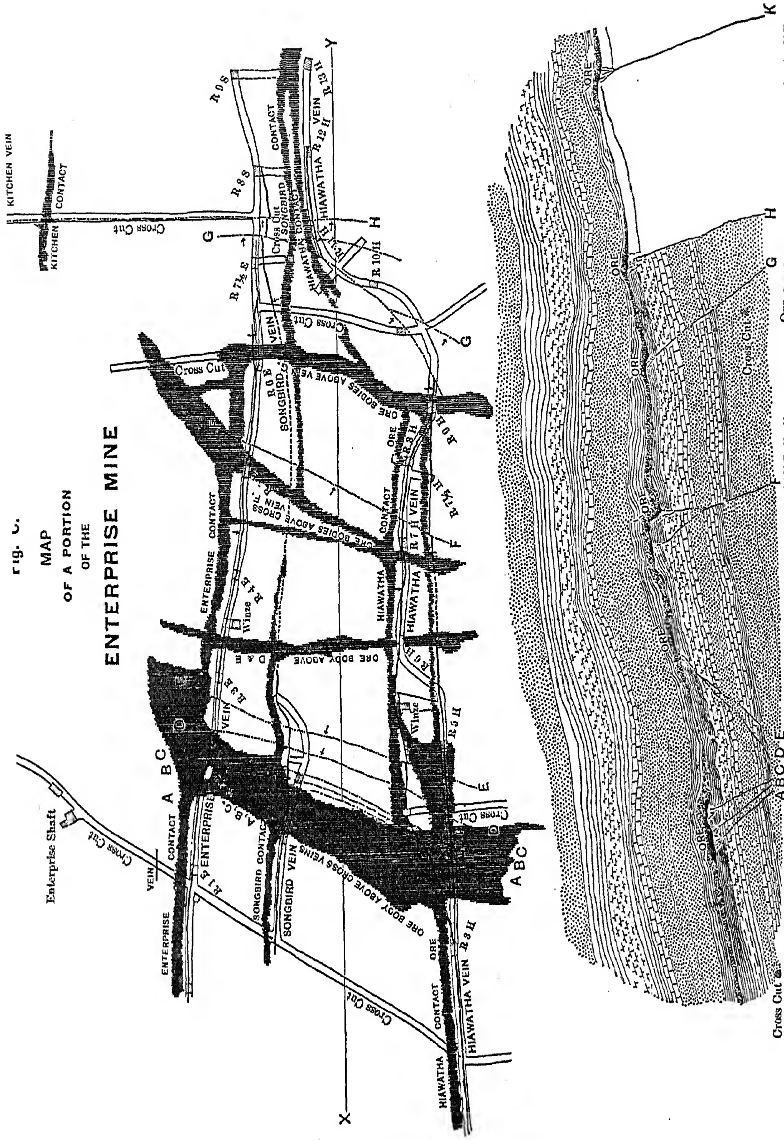
There is too frequent a tendency in mining to look upon veins as necessarily continuous, and to make the nomenclature correspond to the drifts. In this case, a level follows three distinct veins in different parts of its length. The crosscut between R7½E and R8S was put out to search for the Enterprise vein in case it had been faulted by the two cross-veins, G and H. It proved that this particular vein had ceased; but it led to the discovery of the Kitchen lode.

The numerous cross-veins are mapped, and their dislocating influence on the pay-veins is easily discernible.

* A large body of gypsum also occurs along the contact in the Vestal workings of the Rico-Aspen mine.

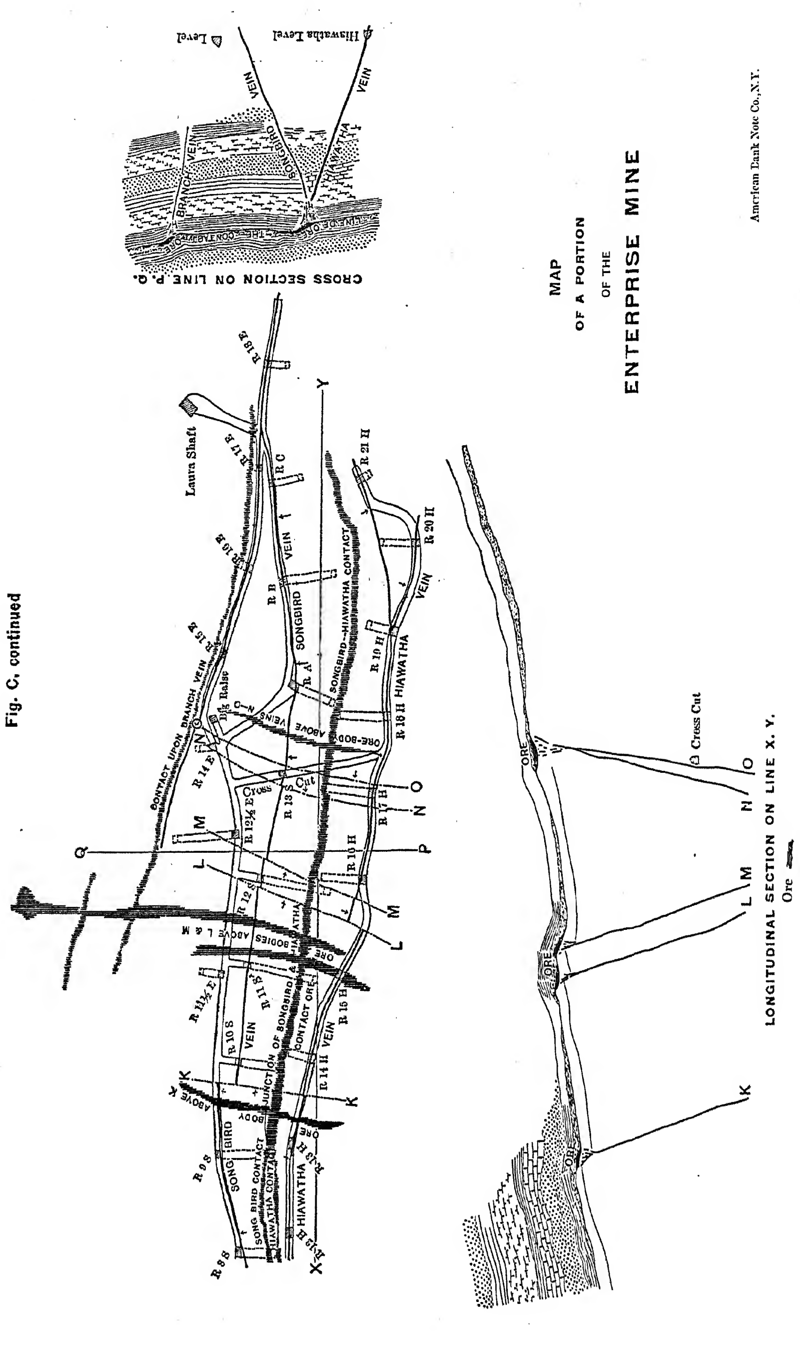
† The raises are numbered and initialed, so that R6E means Raise No. 6 on the Enterprise, R10H, Raise No. 10 on the Hiawatha, etc.

Returning to the crosscut from the Enterprise shaft, we will follow the Songbird, which there appears as a small vein having



a slight easterly dip. Going northward (to the right), the level shows it to flatten. The next time it is seen, 560 feet further

Fig. C, continued



north, it has changed its dip strongly westward. The cross-veins, G and H, fault it, and it appears in the Enterprise drift at R8S. The cross-vein K throws it 19 feet eastward, and it is then seen at intervals in cross-cuts and raises until, beyond R17E, at the crosscut to the Laura shaft, it merges with the branch-vein, sometimes labelled the Enterprise. The Hiawatha is disturbed by the same series of cross-veins, and in one case suffers a very serious dislocation, namely, between R9H and R11H, as already described in the discussion of Fig. 23.

These veins were followed by raises and stopes to the contact overhead, and upon the contact ore-bodies were found having a narrow width and courses corresponding to the strike not only of the pay-veins, but also of the cross-veins. Thus both series of veins, older and younger, rich and poor, are topped by bands of ore distributed along this horizon, so as to make a network the intricacy of which for a long time obscured its real character. The map exhibits the contact ore-bodies and proves very clearly their connection with the two series of veins. The two sections, along XY and PQ, will further help to explain this.

Thus it became evident, after careful surveys and the projecting of the dip of the veins to their intersection with the contact, that no ore occurred upon that contact which was not related to the underlying veins, and that the latter, conversely, were always topped by ore, although that ore was not necessarily always wide enough and rich enough to exploit.

The geological relationship was abundantly proved; and when the writer prepared this map by first putting the veins as the surveys had traced them, and then platting the ore-bodies of the contact as the stopes had exposed them, it was very remarkable to discover how the latter corresponded with the projections of the former.

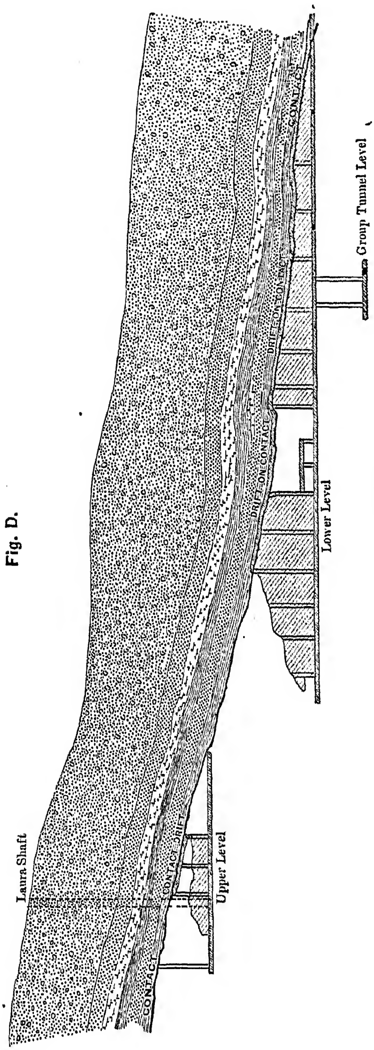
Fig. D is a longitudinal section along one of the pay-veins, showing how the ore ceases at the contact, and how the stopes extend only a short depth below.

Fig. E is a section along the Group tunnel, affording an illustration of the series of pay-veins and their relation to the contact.

VII.—Origin of the Ore-Deposits.

The structure and composition of the ore-deposits of New-

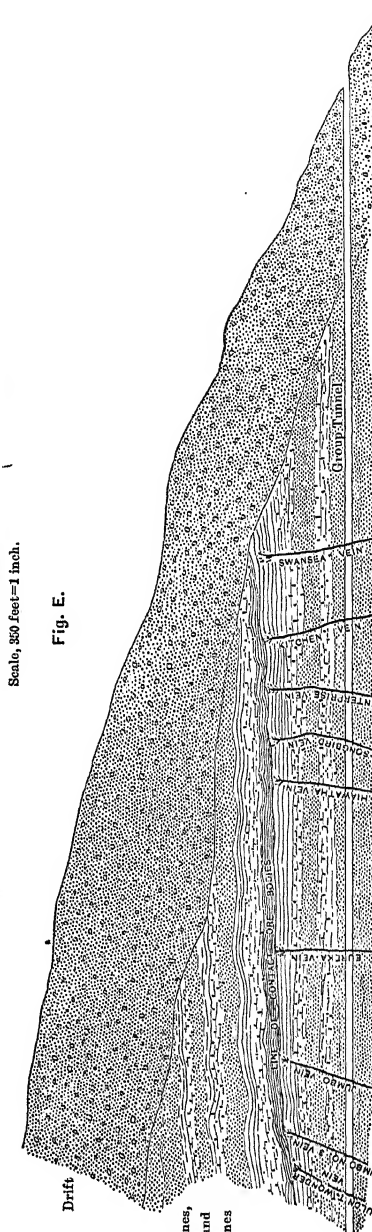
Fig. D.



SECTION ALONG HIAWATHA VEIN.

Scale, 350 feet=1 inch.

Fig. E.



VERTICAL CROSS-SECTION OF THE VEIN-SYSTEM ON THE LINE OF GROUP TUNNEL.

American Bank Note Co., N.Y.

man hill offer suggestions of their origin. Reference has already been made to the slow recognition of their true relations which came in the wake of extended mine-exploration. When the connection between the flat ore-bodies of the contact and those of the vertical veins underneath them was first traced, there arose the idea that the ore of the former had "spilled over" into the latter; in other words, the theory of descending solutions was advocated. Such ideas were expressed by some of the experts who testified during the long litigation between the Enterprise and Rico-Aspen companies. All the available evidence on the subject, however, both geological and chemical, is opposed to this view.

The rocks enclosing the ore-deposits have undergone successive rupturing, resulting in the creation of a series of fractures which have served as water-ways available for the circulation of mineral-bearing solutions. The fact that the ore-bearing verticals penetrate the porphyry, and the shattered condition of the latter along the horizon of the contact, prove that its intrusion among the sedimentary rocks preceded the formation of the ore-bearing fissures. The crossings of the later systems of fissuring establish their relative age. Thus, then, we have evidence that a condition of strain culminated in a multiple fracturing of the Carboniferous rocks, and the formation of certain of these fractures was accompanied *pari passu* by the slow upwelling of mobile igneous matter which, when cooled and solidified, became the porphyrite of to-day. Outside the area of the mine-workings large faults and enormous intrusions of porphyrite did occur, but within the region of ore-deposition the forces at work produced a system of small multiple fractures and did not permit the invasion of large masses of porphyry. These results are vaguely ascribed to a condition of strain, and the analogy of other better known regions permits us to ascribe this strain to a local manifestation of that wider phenomenon called the continental uplift, itself a part of the readjustment of the earth's exterior to a shrinking interior.

During the next disturbance of equilibrium, a disturbance probably due to the contraction following upon the cooling of the included masses of porphyrite, a set of new fractures was formed, and along these underground waters began to move. When they had precipitated valuable ores along their channels

of circulation, later movements produced a series of cross-fractures which faulted them. Minor shiftings, which have supervened at various more recent times, have caused displacements along the bedding, affecting both of the older vein-systems.

The contact-zone has been the victim of all these disturbances. This is to be ascribed directly to its structure. A thickness of closely-laminated shales is laid upon blocky limestones and sandstones. To the formation of a fracture the latter rocks would offer no particular obstacle, because of their homogeneity; but the upward extension of a fracture would be impeded, if not stopped, by meeting a series of beds which, on account of their laminated structure, are easy to bend, but hard to break. There is nothing fanciful in this reasoning. The section given in Fig. 19 affords an illustration exactly in point. Thus, it seems to me, the structure of the rocks of the horizon now known as the "contact," was the immediate cause of the repeated shattering which that horizon underwent; it was the factor which stopped the upward extension of the vein-fractures and produced the consequent limitation to the circulation of those mineral solutions which were the immediate agents of ore-deposition. Thus is explained the concentration of large masses of ore along this zone, because it became a dam, checking the circulation in an upward direction. The fractures now followed by the pay-veins were unable to break through the shales above the contact, and though the later cross-veins were stronger, they too were stopped by the elasticity of these closely-laminated beds.

In each case, therefore, the force of vertical fracturing was diverted into a horizontal displacement which soon made the zone under the shales a mass of shattered rock, peculiarly adapted to become the place of ore-deposition.

The ore is confined to the pay-veins for a depth of 100 to 175 feet below the contact. It does not occur in the cross-veins, but is found in the contact immediately above them, as above the pay-veins. This distribution cannot be satisfactorily explained with positive certainty. There are so many conditions determining the character of an ore-deposit, and ordinary mine-exploration, being a commercial enterprise and not a scientific inquiry, reveals so few of them, that the geologist, though aided by the chemist, is often at a loss. Yet, in this case, the avail-

able evidence, though in many ways inconclusive, is highly suggestive.

The ore-bodies of the contact apparently owe their existence to a combination of chemical and physical conditions. The ending of the vein-fractures at the contact-horizon put a stop to the upward flow of the metal-bearing solutions, and caused them to permeate the shattered rock and distribute themselves along the strike of the veins which had been their passage-way. At the contact they found the chemical precipitant which compelled the dissolved metals to separate out as aggregates of ore. That precipitant was probably the graphite of the black shales, as is indicated by actual experiments presently to be described.

The non-persistence of the ore of the pay-veins below a certain distance from the contact is apparently connected with the circumstance that the sedimentary rocks immediately below the contact are black, by reason of the carbonaceous residues of the vegetation imbedded amid the sand and mud on the floor of an estuary of the Carboniferous period. This carbonaceous matter probably acted as a precipitant of the metal-bearing solutions. As the depth below the contact increases, the rocks lose their blackness, and presumably, therefore, contain no precipitant carbon.

Although the pay-ore terminates at a depth fairly uniform among the different veins, it must not be supposed that the veins themselves cease at this horizon. On the contrary, the fractures maintain their course to depths far beyond the deepest mine-workings; but they become barren of valuable ore, enclosing nothing but quartz and crushed country. The rhodochrosite ceases. The Lexington tunnel, for example, cuts through the lodes which have yielded so richly in the Enterprise workings, 400 feet overhead, and discloses them as veins of white quartz, traversing light gray, coarse-grained sandstone. Both lodė and country-rock have changed in character entirely.

The fact that the cross-veins are barren, and yet rich ore-bodies overlie them in the contact-zone, indicates that such bodies are due to special shattering of the ground by the cross-veins, which has furnished favorable places for ore-deposition. Moreover, barren as the cross-veins are, they appear to influence the richness of the pay-veins. It is common to find ores of more than average grade in the pay-veins, where they are

broken by the cross-veins. But the mineral solutions apparently did not rise through the cross-veins. They must have circulated along the verticals; and the deposition of ore in connection with the cross-veins, as noted, may have been not only a collateral, but even a secondary process. The superior richness of these bodies in many cases may indicate that they have resulted from a re-solution and re-deposition of the contents of the verticals.

The idea of the precipitation of the ore through the agency of carbonaceous matter has been advanced in connection with ore-deposits in other regions. I may quote as instances the black Silurian slates of Bendigo, Victoria;* the Devonian slates of Gympie, Queensland; the Jurassic slates of the "mother lode" region in Calaveras and Amador counties, California; the black shale enclosing the gold-specimen ores of Farncomb Hill, Breckenridge, Summit county, Colorado; the graphitic casing occasionally seen in the ores of the Sunnyside and Mastodon veins, in San Juan county, Colorado; and the celebrated Indicator† series of Ballarat, Victoria.

In order to test this theory, I broke some pieces of black shale on the contact above Raise 12 on the Enterprise vein, and took them to the Argo smelter, where, by the kindness of Mr. Pearce, the following experiments were made. A piece of the Rico shale was put into a weak solution of sulphate of silver (Ag_2SO_4) containing some free acid intended to neutralize the lime ($CaCO_3$) in the shale. The precipitation of metallic silver became visible in three days. The parallel experiment with gold was more interesting. A piece of ore (assaying 1147 ounces of gold per ton) obtained from the Prince Albert mine at Cripple Creek was taken, and its gold was extracted by a solution containing ferric sulphate ($Fe_2O_33SO_3$), common salt (NaCl) and a little free acid (H_2SO_4). This Cripple Creek ore carried the black oxide of manganese (MnO_2) in visible quantity, and thus the chlorine used to form the gold-solution was liberated in a manner simulating natural conditions. Of the gold in this Cripple creek ore, 99.91 per cent.

* Discussed by the writer in *Trans.*, xxii., 319 *et seq.* Also by Mr. Argall, same volume, p. 762.

† See also "The Indicator Veins, Ballarat," by the writer, *Eng. & Min. Jour.*, Dec. 14, 1895.

was extracted, and subsequently precipitated on the Rico shale by inserting the latter in the solution thus formed. The gilding of the black shale by the deposit of gold became visible within four hours.

The order of succession of the various minerals composing the ore is indicated in many ways. Beautiful pseudomorphs of quartz after baryta have been found. The replacement of rhodochrosite by quartz is often discernible. That the baser sulphides frequently enclose fragments of rhodochrosite (Fig. 25), establishes their relative age. Pieces of country found within the vein-matter are occasionally surrounded by a rim of rhodochrosite. (See Fig. 34.) The silver sulphides, and the native gold associated with them, occur exclusively within the geodes which are usually distributed along the center of the vein. The veins enclose shreds of country-rock; sometimes such pieces of rock are found within masses of sulphide ore, and there is an imperceptible gradation from clean sulphide to rock so impregnated with ore as to have its true character obscured. Banded structure is common.

This succession points to the following conclusions: When the fractures were first formed they consisted of lines of crushed country, afterwards healed by a deposit of carbonate of manganese. The latter (rhodochrosite) is likely to have been derived from limestones occurring at a horizon not necessarily very far below the place of the present ore-deposits. Then came a fresh fracturing, accompanied by the deposition of baryta and the sulphides of lead and zinc. Later still the earlier vein-stuff became shattered by fissuring on the old lines of movement, and along the water-way thus created there came siliceous solutions, which replaced baryta and rhodochrosites with crystalline quartz. Finally, the vein was riven along its center, and waters rich in the salts of gold and silver found their way upward to undergo precipitation through the agency of the rhodochrosite and the shattered portions of the carbonaceous country enclosed within the vein-walls.

We are driven in this case to the hypothesis of ascending solutions. A lateral flow must be a part of an upward or downward movement of the underground circulation. As a general phenomenon it is inconceivable. The deposition of ore from descending solutions is in this case chemically possible through

the reduction of sulphates by carbonaceous matter. But for the hypothesis of descending sulphates there is no basis of fact. The geological evidence is all against it. The structure and environment of the ore-bodies point to their derivation from solutions which came up from below. The passage-ways open to the circulating waters cease upward and extend downward; they connect with no available origin in one direction, but lead to a possible source in the other.

The Invention of the Bessemer Process.

BY JOSEPH D. WEEKS, PITTSBURGH, PA.

(Presidential Address at the Pittsburgh Meeting, February, 1896.)

Note by the Secretary.—This address having been made the object of much hostile comment, arising, as Mr. Weeks believed, in large part from misunderstanding of its purpose and meaning, was, by his express direction, withheld from official publication in the *Transactions* until he should have so modified or added to it as to make such misunderstanding impossible. His illness and death prevented him from executing this intention; and, consequently, the only version of the address now in the Secretary's hands is one of which the author had forbidden the publication. Under the circumstances, therefore, it seems best to publish, in this place, only some portions of the address and its appendixes which have a historical value and are beyond controversy, giving at the same time a summary of its general nature, based upon the original manuscript and upon correspondence with Mr. Weeks. For this summary, the Secretary is alone responsible.

The address was essentially a restatement of well-known and established facts, which it interpreted as proving that, of the three elements of the Bessemer process (namely, the pneumatic principle, the successful mechanical application of that principle, and the recarburization with spiegeleisen), William Kelly was the first inventor of the first, Bessemer of the second, and Mushet of the third. Mr. Weeks used the words "original inventor" in the sense (as he explained in one place) of *first* inventor, and neither asserted nor believed that Bessemer's con-

ception of the pneumatic principle was not original in the sense of being *independent*. The *priority* of Kelly's invention was his only contention; and as this had been repeatedly asserted,* and had been established by judicial investigation,† he did not expect it to be contradicted. Undoubtedly this was the chief point which he wished to make plainer by modifying the address before its official publication. Another point, I believe, was the fact that Kelly, though he proposed to decarburize and refine liquid iron "without the use of fuel" by blowing air through it, did not in fact carry out this principle so as to get a final liquid product. In other words, his bath "came to nature," like that of a puddling-furnace. This fact appears in the address in one or two places—for instance, in the following passage:

"As Mr. Holley puts it: 'Mr. Bessemer's invention consisted in the mechanical means for intimately diffusing the air throughout the mass of iron to such an extent that the chemical changes take place with sufficient rapidity to leave the mass fluid after the carbon and silicon have been removed; while the means employed by Mr. Kelly were not adequate to carry the process to the same extent.'" (See Appendix VIII.)

It also follows closely from the description of Kelly's Johnstown converter of 1856, given to Mr. Weeks by Mr. John E. Fry:

"The apparatus was a 2-foot section of a 3-foot boiler-shell lined up to about 16 inches inside diameter with fire-brick, having a paved fire-brick bottom, about 20 inches deep. This was put just outside of the door of the blast-engine room of the old Cambria blast-furnace. The engines gave from 3 to 4 pounds blast-pressure. There was a temporary connection made from the blast-pipe to this little auxiliary furnace by a gas-pipe. The nozzle was a cast-iron affair, such as was used at the tuyeres in those days. The tuyere was "clayed up" and dried and connected by a loose elbow to the pipe, so as to be swung down into the metal. The metal was melted at the foundry, and brought up in a ladle containing about 500 pounds. The furnace was about 200 yards from the foundry, and the ladle was put on a little metal-hauling car and hauled to the converter. This metal would make probably 8 or 9 inches of depth in this little pot. As soon as the metal was dumped in the pipe was shoved down with the blast on. A cover of pieces of sheet-iron was laid across the top to prevent the sparks flying too freely."

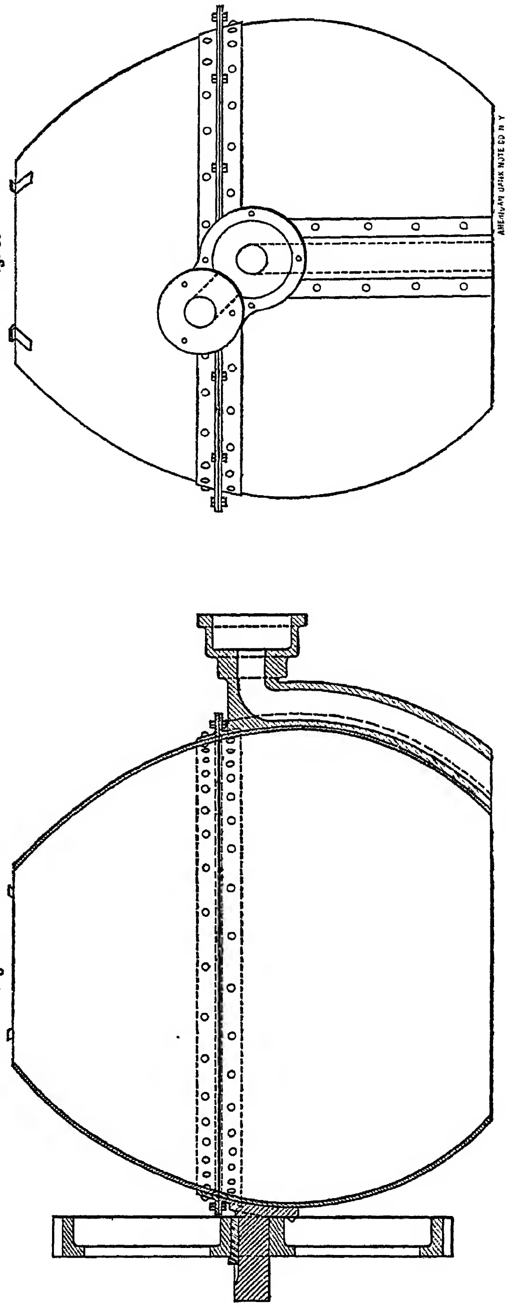
Finally, the same fact as to the difference between Kelly's actual result and that of Bessemer was brought out in the oral

* See Mr. Robert W. Hunt's "History of the Bessemer Manufacture in America," *Trans*, v. 201; also, Swank's *Iron in All Ages*, 2d ed., p. 395. I am not aware that these publications aroused special protest in any quarter.

† See Appendixes, VII. and IX., below.

Fig. 5.

Fig. 4.



Converter used at Johnstown, Pa., in 1861.

discussion of the address at Pittsburgh. I believe that Mr. Weeks, in revising the address, would have made it still plainer that he admitted this difference, while he would doubtless have continued to adhere to his view that it ought not to deprive Kelly of the credit of the first discovery of the "pneumatic principle."

Concerning Bessemer's first converter (see Appendix X), Mr. Weeks remarked that it was also fixed, but "a vastly different piece of apparatus from the Kelly converter. Moreover, the records of the Patent Office show that Bessemer very soon discarded this for the movable rotary tilting converter, which is practically the vessel still used."

The first rotary converter in the United States (see Figs. 4 and 5) was erected at Cambria in 1861. Mr. Weeks says it "was an infringement on Bessemer's converter."

The chief object of his address was to present interesting documents and details, which are contained in the Appendixes, all of which are here given.

APPENDIX I.

Interference File—Wm. Kelly vs. Henry Bessemer.

Deposition of S. I. Smith.

State of Kentucky, Lyon County. } ss.:

My name is Strother I. Smith. I am a constructor and builder of blast-furnaces and machinist in general. My family reside in Eddyville, Ky., and I am now engaged as machinist at the Cumberland Iron Works, in Stewart county, Tenn. In the year eighteen hundred and forty seven (1847) occurred the remarkable flood in the Cumberland river, which enables me to fix the date in my recollection of the facts I am about to state. I lived in the neighborhood of Eddyville, Ky., at the time, and was at the forge of Kelly & Co., in Caldwell, now Lyon county, Ky. Whilst there William Kelly gave me a description of a new process for treating iron, by which he expected to be able to produce malleable iron from fluid pig-metal without fuel, by a strong blast of air blown into the liquid mass of iron. My recollection on the subject is clear and perfect, as detailed to me by William Kelly at the time by drawings and descriptions. His theory was this, that by blowing a blast of air into a body of liquid iron, the oxygen in the air would combine with the carbon in the iron, and in so doing a heat would be created in the iron of sufficient intensity to carry it through the operations of refining and bringing the iron to nature. From the description Mr. Kelly gave me at the time of his improved process, I could readily have carried it out, if so desired, by the instructions he gave me then, and put it in practical operation, at the aforesaid date, say 1847. S. I. Smith.

To this affidavit is attached a jurat, dated 16th March, 1857.

APPENDIX II.

Affidavit of Jeremiah Tiley.

STATE OF KENTUCKY, } ss.:
LYON COUNTY. }

My name is Jeremiah Tiley. I am a forgeman of about 35 years' experience. I reside in Pope county, Ill.

I was in the employment of Kelly & Co., as forgeman at their Union Forge, in 1847; during this year William Kelly told the forgemen that he could make wrought-iron without fuel out of liquid pig-iron in a crude state. His theory was that by blowing blasts of air into a liquid body of metal, that instead of chilling up the iron, as most persons said he would do, a high heat would be generated by the oxygen in the blast combining with the carbon of the iron, thereby rendering fuel unnecessary. Said Kelly made several large drawings and put them in the forge, that all the forgemen might see and understand his plans. I saw the drawings and also saw the furnace he built in 1847 to carry out his new process. I also saw two furnaces he built at Suwanee Iron Works in 1851, which were used for a long time; the process was known here as Kelly's Air Boiling Process. I made an engagement with said Kelly to stay at the furnace for a week and carry on his process for him; was prevented from going. Saw Mr. Kelly break a piece off of a piece of iron taken out of the air boiling furnace and take it to the blacksmith's shop and heard him hammering at it. He returned with it and showed it to his brother as some of the iron he made by his new process; it looked like good wrought iron.

his
JEREMIAH + TILEY.
mark.

Attest:

C. M. SHELLEY.

To this affidavit is attached a jurat, dated 16th day of March, 1857.

APPENDIX III.

Affidavit of Alfred H. Champion.

STATE OF KENTUCKY, } ss.:
LYON COUNTY. }

My name is Alfred H. Champion. I am a physician, and reside in the town of Eddyville, Lyon county, Kentucky.

In the fall of 1851 I was present, in company with William Kelly, some two or three practical iron-masters and others, when William Kelly informed the company there assembled, that he had just finished a new furnace for making wrought-iron without fuel and invited all present to go to his iron-works and see it in operation. This being a subject of great interest with the iron-masters then present, they asked him what his process was like. He described it thus: His furnace was something like the common cupola-furnace, into which he would put a quantity of fluid metal taken from his blast-furnace, and blow blasts of air into the liquid iron, without any fuel, when it would commence boiling, and continue to boil until, like the puddling-fire, the iron would be boiled to nature or made malleable.

This announcement by Mr. Kelly was at once condemned by all the iron-men present, who were of the opinion that he would immediately chill up his furnace. Said Kelly then went into further explanation of his theory, and told the company that instead of chilling up his furnace, it would become intensely hot, that the car-

bon contained in the iron, having an affinity for oxygen, would unite with the oxygen in the blast and so united would create an intense heat and pass off in the form of carbonic acid gas. The company present all differed in opinion from Mr. Kelly, and appealed to me as a chemist in confirmation of their doubts. I at once decided that Mr. Kelly was correct in his theory, and then went on to explain the received opinion of chemists a century ago on this subject, and the present received opinion, which was in direct confirmation of the novel theory of Mr. Kelly. I also mentioned the analogy of said Kelly's process in decarbonizing iron to the process of decarbonizing blood in the human lungs. A. H. Champion, M.D.

A jurat is hereto attached, dated 14th day of March, 1857.

APPENDIX IV.

Affidavit of Charles C. Cargill.

State of Kentucky, } ss.:
Lyon County. }

My name is Charles C. Cargill. I am a founder by profession, and have a knowledge of chemistry, at least so far as relates to the operation of making iron is concerned. I reside in Stuart county, Tenn.

In the fall of 1851 I was founder at Suwanee Iron Works, owned by Kelly & Co. I left the employment of said Kelly & Co. in November, 1851.

Whilst in the employment of said Kelly & Co., William Kelly described to me a novel mode of treating iron, which he claimed to have discovered some years previous. It was for converting crude pig-iron into malleable-iron, without fuel, by blowing blasts of air into the melted iron. His theory was that the carbon contained in the iron, having an affinity for oxygen, would unite with the oxygen in the blast and cause a boiling to take place in the iron, which, after being continued, must bring the iron to nature, without any other manipulation. He also told me that this chemical combination of carbon and oxygen would put intense heat into the iron. I differed in opinion with Mr Kelly. I also asked him how he was to get a fiber in the iron. His answer was that the action of the blast would have a mechanical as well as chemical effect on the iron that would produce a fibre. William Kelly built, whilst I was at Suwanee Iron Works, a small cupola or furnace, for the purpose of putting his new process in practical operation. It was much like a common cupola furnace, and had a tuyere in the side entering the chamber near the bottom.

The blast was first let on to the tuyere, and whilst blowing the liquid iron was poured into the furnace. I was often present and saw the said process in practical operation. I remember seeing some of the iron so treated, but do not now remember the exact quality of the iron. I do know it was not crude pig iron, and think it was somewhat malleable, or partook largely of the properties of wrought-iron.

Chas. C. Cargill.

A jurat is hereto attached, dated 12th day of March, 1857.

APPENDIX V.

Affidavit of William R. Edes.

State of Kentucky, } ss.:
Lyon County. }

My name is William R. Edes. I am a machinist by profession and a licensed preacher of the Gospel in the Methodist Episcopal Church South. I reside in Lyon county, Kentucky.

In the year 1851 I was in the employment of Kelly & Co. as machinist at their iron-works in Lyon county, Ky. I made for William Kelly some pipe-patterns, the castings off of which were intended to be used about a furnace which William Kelly was then (1851) building, and which I rather think I helped to build also. The pipes were cast and applied to said furnace. The object of said furnace was to carry out an improvement of said William Kelly by which he told me he could make wrought-iron without any fuel, that he could, by blowing blasts of air into the liquid iron, boil the iron and bring it to nature. When the furnace was ready William Kelly told me he was going to put it in operation and requested me to come and see it. I went and saw it at work. The blast was first let on to the furnace, and then the iron was afterwards poured in. William Soden, Washington Jefferson and others took the liquid iron as it flowed from the blast-furnace and ladled it into the boiling-furnace. After this experiment I was often present whilst William Kelly was blowing his furnaces. I remember having seen two such air-boiling furnaces—one a hot, the other a cold blast. I saw some of the iron made by Mr. Kelly with his air-boiling furnace, and thought at the time that it was good wrought-iron. I conversed with a blacksmith who told me he had worked some of the iron made by Kelly's new process (this was in 1851); he represented it to me as first-rate iron.

The said air-boiling furnaces that I saw were built circular, a good deal like in the inside unto a moulder's standing ladle, with a bottom shaped like a basin, each had a tuyere in the side entering near the bottom; they were enclosed at top, with a vent for the escape gases.

I also remember having heard of a furnace built by said Kelly some years prior to 1851, for the same process as the above. I did not see it, however.

WILLIAM R. EDES.

A jurat is hereto attached, dated 1!th March, 1857.

APPENDIX VI.

Affidavit of William Soden.

STATE OF KENTUCKY, } ss.
LYON COUNTY. }

My name is William Soden. I am a moulder by profession, and reside on my farm in Lyon county. In the month of November, 1851, I moulded and cast for William Kelly at the "Suwanee Furnace," in this (Lyon) county, a set of blast pipes, which he applied in that month and year (1851) to a small cupola or refining-furnace, known at the above establishment as an "air-boiling furnace."

I know the date well, as I keep a daily account of all the work I do at the above works, and I find those pipes were made by me in November, 1851. In addition to my own account, I find upon examination that the said pipes or castings are placed to my credit in the same month and year (November, 1851) on the books of the furnace. I also know the date from another reliable source to be November, 18 ,1. The said pipes were put in use during November, 1851, by William Kelly to a cupola or "air boiling" furnace constructed in the following manner: A cylindrical chamber covered at the top with a vent to carry off the escape gases. The bottom of said chamber was concave, and near the shape of the point of an egg. In the side of and near the bottom was fixed a tuyere. The blast of air was first let on to the furnace and the liquid metal taken from the blast-furnace and

poured into said "air-boiling" apparatus. The blast of air soon put the mass of liquid metal into a boiling commotion, thereby freeing the metal of its impurities which were disengaged by the escape of gas and cinder. Mr. William Kelly's theory, as explained to me by him at the time, was that by blowing a strong blast of air up and into a mass of liquid crude metal the carbon contained in the iron would unite with the oxygen in the air and pass off in carbonic acid gas, by which the iron would be decarbonized, and by the boiling it would thereby receive be worked "to nature" or malleable iron without the aid of fuel.

I was often present when he had his "air-boiling" furnace in operation, and frequently assisted in ladling the melted metal during the aforesaid month and year (November, 1851) and since from the blast-furnace, and poured it into his "air-boiling" furnace.

I have seen said William Kelly use the cold and hot blast in his experiments and trials, and know of seven different air-boiling furnaces that he has built during and since 1851, which were all alike in principle. Some had one, some four and some eight tuyeres. The action of all were alike and the principle all the same.

WILLIAM SODEN.

A jurat is hereto attached, dated 12th day of March, 1857.

APPENDIX VII.

Decision of the U. S. Patent Office.

U. S. PATENT OFFICE,
April 13, 1857.

In the matter of interference between the patent of Henry Bessemer, of London, and the application of William Kelly, of Lyon county, Ky., for Improvement in the Manufacture of Iron and Steel, the hearing of which was fixed for the first Monday in April:

It appears, that by the concurrent testimony of numerous witnesses, Kelly made this invention and showed it by drawings and experiments as early as 1847, and this testimony appears to be reliable in every respect.

The patent of Bessemer was sealed at London on the 11th of April, 1856, and bears date 11th October, 1855.

Priority of invention in this case is awarded to said Kelly, and it is ordered that a patent be issued accordingly, unless an appeal be taken within sixty days from this date.

S. T. SHUGART,
Acting Commissioner.

APPENDIX VIII.

Evidence of A. L. Holley.

In the matter of the application of William Kelly, for an Extension of Letters Patent for an Improvement in the Manufacture of Iron, No. 17,628, dated June 23d, 1857. Reissued November 3d, 1857. No. 505.

Alexander L. Holley, a witness for the applicant herein, being duly called,

cautioned and sworn, testifies as follows, in answer to interrogatories proposed by L. S. Durfee, Esq., representing counsel for applicant.

1. What is your name, age, residence and occupation?

A. Alexander L. Holley. I am 38 years of age. I reside at Brooklyn, King's county, N. Y. I am an engineer and manager of Bessemer Steel Works at Troy, N. Y.

2. Have you read the letters-patent and reissued letters-patent of William Kelly?

A. I have.

3. Will you please state what you consider the essential features of said patent?

A. The essential feature of the patent is the decarburization of crude cast-iron by the air-blast in a vessel independent from the blast-furnace or furnace in which it was melted and without the application of external heat.

4. Have you read the depositions in the interference between Kelly and Bessemer at the time of Kelly's application for his patent?

A. I have.

5. Was the process as described by Kelly in his patent and by the witnesses in his behalf in such interference new and useful?

A. As far as I am aware it was new. I know it to be useful.

6. What are the relations of the process described by Kelly to what is generally known as the pneumatic process for the manufacture of steel?

A. Chemically the two processes are identical, and mechanically the principle is the same, and the means are the same in general character but not in detail.

7. What is the chief difference between the process as described by Kelly and that now used?

A. The strength and distribution of the blast are greater in the present process. Mr. Bessemer's experiments in this matter were nearly contemporaneous with those of Kelly, and Mr. Bessemer's invention consisted in the mechanical means for intimately diffusing the air throughout the mass of iron to such an extent that the chemical changes take place with sufficient rapidity to leave the mass fluid after the carbon and silicon have been removed; while the means employed by Mr. Kelly were not adequate to carry the process to the same extent.

8. Then the difference between the process of Mr. Kelly and that of Bessemer was principally one of degree?

A. Yes.

9. What do you consider the value of the invention of Mr. Kelly in its relation to the pneumatic or Bessemer process as at present practiced?

A. I consider Kelly's invention the first practical development of the pneumatic process, and it has been so recognized by the owners of the combined patents covering this process. Mr. Bessemer, by his superior mechanical improvements and capital, anticipated the results which might naturally have followed from the development of the Kelly process; but Mr. Kelly's invention was valuable in the direction in which he applied it.

10. What do you consider the economy developed by Mr. Kelly's invention in the mode in which he practiced it?

A. The process applied by Mr. Kelly developed a considerable economy in the fuel required to melt the iron and to keep it fluid. Judging from the Kelly testimony, before referred to, I think the saving in fuel, by his process, would average for the works in this country, not less than one dollar per ton of iron treated.

ALEXANDER L. HOLLEY.

APPENDIX IX.

Report of the Examiner.

ROOM No. 151, U. S. PATENT OFFICE,
June 7, 1871.

To the Commissioner of Patents:

SIR: In the matter of the application of William Kelly for extension of patent for "Improvement in the Manufacture of Iron," issued to him, and dated June 23, 1857, and reissued November 3, 1857 (No. 505), I have the honor to submit the following report:

The invention in this case consists in decarbonizing molten crude cast-iron by running it into a vessel separate from that in which it was melted and blowing through it blasts of air, so as to burn out the excess of carbon and refine the metal.

The application on which this patent was allowed was filed November 13, 1856, and put into interference with a patent then just issued to Henry Bessemer, of England. I am not familiar with the facts developed in that interference, but the result was that so far as it concerned the particular matter claimed by Kelly, priority of invention was awarded to him.

With the exception of the Bessemer patent, just referred to, there was nothing that would anticipate Kelly's claim.

Christian Shunk had filed an application for a patent in 1854, but showed no method of applying an air-blast to molten pig metal except in the hearth of the blast-furnace.

Martien had also, in this country and in England, blown a current of air through a stream of cast-iron as it ran down the spout from the blast-furnace to the "pig-bed." But neither of these was an equivalent for the method of Kelly of running the molten iron into a vessel separate from that in which it was melted, and passing blasts of air through it until refined, or more or less decarbonized and dispensing with other fuel

The invention was therefore new and patentable at the time of the grant of the original patent

The inventor used diligence in introducing his invention, but was not successful, for the reason that the working of the invention required a large outlay of capital, and the conflicting claims of others prevented any concentration of effort for the development of either invention until within the last few years, since when the several interests have worked in harmony.

In regard to the utility of the invention of Kelly, the testimony is conflicting. The witnesses Holley, Durfee and Thompson, who are very familiar with all that has been done in the pneumatic process of treating iron, testify to the utility and importance of the invention.

On the other side are the witnesses Wright and Kimball, who have also had considerable experience in this process

Exception is taken to the receipt of the testimony of Wright and Kimball, for the reason that the notice was not served in time. The objection seems valid.

On the part of Kelly, the witness Lynn states that he witnessed Kelly's experiments in 1851, and that these experiments were continued until 1856; that Kelly's method was economical and useful, effecting a great saving in both fuel and labor over the finery and run-out.

The witness Edes, who is well acquainted with iron manufacture, saw Kelly's early experiments, and fully corroborates the statement of Lynn in regard to utility and economy of the process.

This witness estimates a saving of $7.00 per ton of iron.

Kelly's own statement of the history of his invention is full and clear, and, when taken in connection with the statement of the witnesses, seems to be both intelligent and truthful.

In the matter of expenditure, applicant makes an account of $61,500, but this embraces the sum of $50,000 for "loss in business"

The receipts from the patent amounted to only $2400.

Opposition to the grant of the extension has been regularly put in by S. W. Kirk and C. Shunk. The reasons alleged for refusing the extension have not been sustained by proofs, unless, as stated above, the invention is not sufficiently described to be useful.

Shunk refers frequently to a disclaimer in Kelly's patent of January 20, 1857, and this is in no way pertinent to the issues in this case.

The application on which the patent of January 20, 1857, was issued, was filed December 9, 1856, a month and a half later than the application on which the patent of June 23, 1857 (now sought to be extended), issued. The reason for the earlier issue of the later application being the time consumed in the Bessemer interference. The patent of January 20, 1857, is for a different method, perhaps, similar in principle, but not described in the earlier application, on which the patent of June 23, 1857, was allowed.

The subject-matter of the patent of Jannary 20, 1857, is quite similar to, if not identical with, that described in the withdrawn application of C. Shunk, filed August 28, 1854; that is, the iron is treated by an auxiliary tuyere applied at the bottom of the blast-furnace and not in a vessel separate from the furnace in which the iron is melted.

Shunk also refers to his patent of 1859. The patent was issued long after the patent of Kelly, and is for a method of giving a rotary motion to the metal while it is treated with air.

It is for matters not at all involved in this case, and has no more bearing on the subject than any patent for improvement in the pneumatic process.

The above appear to be all the material facts in the case.

Respectfully submitted,

B. S. Hedrick,

Examiner.

APPENDIX X.

Extracts from The Artizan, No. CLXIV., Vol. XIV., London, September 1, 1856.

Bessemer's Process of Manufacturing Malleable Iron and Steel without Fuel.

. . . . The following is a description of the apparatus employed by Mr. Bessemer for the purpose of treating the molten metal according to his process:

Fig. 1 exhibits a vertical section of a cast-iron cupola-like cylinder, the interior being built in fire-brick.

Fig. 2 is an elevation partly in section, exhibiting the construction with greater minuteness, and showing the metal just after it has been poured in and the blast turned on.

Fig. 3 is a sectional plan taken through the tuyeres; *a* is the cylindrical iron casing of the converting-vessel; *b*, the internal lining of fire-bricks; *c*, the lower chamber, into which the metal is poured, and wherein the boiling takes place, *d* being the molten metal; *e* is the upper chamber, above the throat of the domed boiling-chamber, *c*; and into this upper chamber *e*, scraps, gate-pieces and other

metal may be placed, and arranged round the opening, by which means the waste heat from the process of boiling is made available for melting the metal for the next charge; *f* is an annular belt, or air-passage, communicating with the tuyere-pipes, *g* being the tuyere-pipes and nozzles; *h* is the blast-pipe connected with the blast engine; *ii* are openings for the escape of flame and gaseous products, and

Fig. 1.

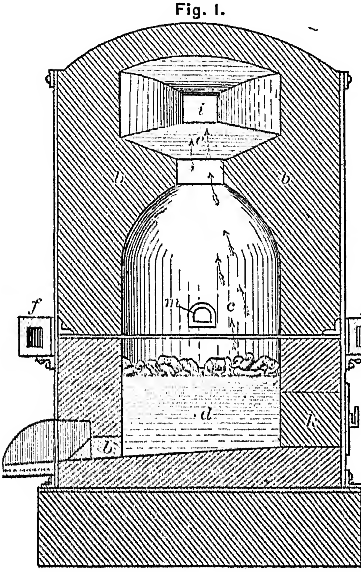
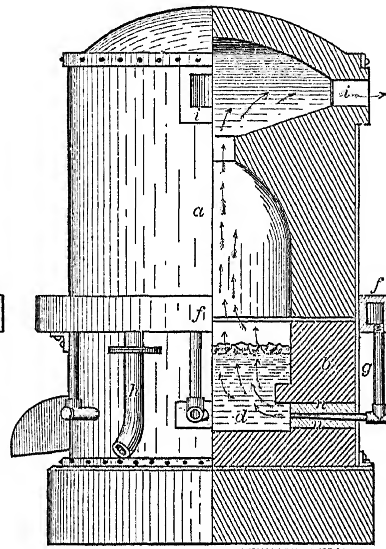
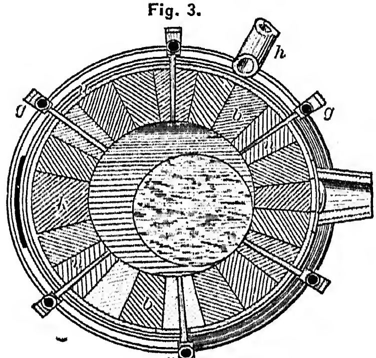


Fig. 2.



AMERICAN BANK NOTE CO.,N.Y.

Fig. 3.



Bessemer's Converter, 1856.

from which also the eruption of slag takes place during the boil; *j* is the tapping-hole; *k*, a man-hole for cleaning out the vessel, and by which the brick lining may be repaired; *m* is the opening through which the crude iron is run into the converting vessel; *n*, *n*, the fire-clay tuyere-blocks, which have been found to answer better than any other arrangement, as they are easily replaceable when burnt away or otherwise damaged. . . . .

An Improved Assay-Muffle.

BY ARTHUR S. DWIGHT, PUEBLO, COLO.

(Colorado Meeting, September, 1896.)

THE accuracy of the silver-assay depends in great measure upon a careful regulation of the heat of the muffle during the process of cupellation. At the beginning of the operation, a relatively high temperature is required to "open" the lead buttons, that is, to clear off the black film of oxide that covers the surface of the metal button immediately after fusion has taken place. The muffle is usually closed up until the buttons are uncovered. As soon as a clear, free surface of molten metal is exposed to the oxidizing action of the air, the temperature should immediately be lowered to the minimum temperature at which the formation and absorption of litharge can progress freely, and the buttons still be kept from "freezing." The process, under satisfactory conditions, will be characterized by the formation of rings of litharge crystals ("feathers," as they are styled), on the cupel, about the oxidizing globule; and the nearer these rings of litharge crystals can be made to approach the central oxidizing globule without interfering with the freedom of oxidation and absorption, the smaller will be the volatilization of silver, and consequently the greater the degree of accuracy attained. This minimum heat must be maintained uniformly during the oxidation of the lead; and just as the last of the lead is disappearing, and the button is preparing to "blick," the temperature of the muffle should be allowed to rise so that the last traces of lead will be driven off. This regulation of the heat calls into play all the skill of the assayer, and requires a vigilant eye, able to distinguish between the slightest variations of temperature of the muffle, assisted by a nice adjustment of fuel and draft. With a few cupels in the muffle at one time, the proper heat for each cupel can be maintained by roughly approximating the proper temperature, and then moving the cupels backward and forward in the muffle; but this is gene-

rally out of the question where hundreds of determinations must be made in a day, and therefore some other method of regulation must be employed. With this end in view, the skill of the assayer will show itself in securing a nearly uniform size of lead button from the preliminary scorification or crucible fusion, so that, when handled in one batch, the buttons will finish cupelling at nearly the same time. The problem then is to lower the temperature of the whole muffle suddenly and uniformly, after the buttons have opened, keep it steady during the cupellation at the minimum temperature, and then suddenly raise it at the end, to secure a "hot blick." The solution is to be found in keeping the fire hotter than is necessary to hold the muffle at the proper degree and cooling the latter by independent agencies, which can be regulated with greater nicety than can the fire itself, and which, when removed, permit the surplus heat of the fire to assert itself, and quickly raise the temperature of the muffle. Ordinarily this is done by placing cold bodies, such as old crucibles and scorifiers, in various positions in the muffle, where they will have the cooling influences desired, even going to the extent of putting here and there, in actual contact with the cupels, a cold scorifier, resting on the raised edges of two or more cupels, so as not to cut off the supply of air. In the hands of a skillful assayer a muffle full of cupels can be manipulated as perfectly as a single row, with every button surrounded with a ring of beautiful litharge feathers.

It is the object of this paper to call attention to an improvement upon this method of temperature-regulation, which has been in use for over a year in the assay-department of the Colorado Smelting Company, at Pueblo, and which was devised by Mr. Howard F. Wierum, the assayer in charge, with the co-operation of Mr. F. L. Capers, President of the Standard Fire Brick Company, of Pueblo, who has shown much intelligence and skill in carrying out the idea. The muffle is moulded with two sets of horizontal ribs on the inner sides, running from front to back. These strengthen the muffle, and at the same time serve to support a loose slab of burned fire-clay about ½-inch thick, with the same width as the muffle, which can be slipped in like a shelf. It was originally hoped to cupel successfully several stories of cupels at the same time, but this is not practicable.

Used simply as a means of heat-regulation, however, the effectiveness of this arrangment is remarkable, and most perfect results in cupellation can be obtained with its aid. When the shelf is slipped over a batch of hot buttons that have just opened, a shade passes over the muffle uniformly, as the temperature suddenly falls. By replacing the heated slabs with cooler ones, the temperature can be kept steady; or if certain rows of cupels are too hot, short strips of slab can be placed exactly over the spots desired. The risk of getting the temperature so low as to "freeze" the buttons can be avoided by placing dummies, or indicators in the front rank of cupels, where the temperature is lowest. When these show signs of freezing, it is an indication that the temperature is getting too low, and proper steps must be taken at once to correct it. By then removing the shelves entirely, the temperature will rise rapidly, if the fire is in proper condition. "The Cathedral Muffle" is the name given to this design, from its fancied resemblance, when in operation, to the interior of an illuminated church.

DISCUSSIONS.

The Physics of Cast-Iron.

Continued Discussion of the paper of William R. Webster. (See Vol. xxv., pages 84 and 964.)

(Pittsburgh Meeting, February, 1896.)

Note by the Secretary.—The paper by Mr. Webster, inaugurating this discussion, was read at the Florida meeting, March, 1895, and a discussion of it was published among the papers of that meeting. To the continued discussion belong the following contributions, issued as separate papers: "Standard Physical Tests for the Product of the Blast-Furnace, and Their Value," and "The Effect of Expansion on Shrinkage and Contraction in Iron Castings," by Thomas D. West, Sharpsville, Pa.; "The Effect of Additions of Titaniferous to Phosphoric Iron-Ores in the Blast-Furnace," by A. J. Rossi, New York City; and "The Mobility of Molecules of Cast-Iron," by Alexander E. Outerbridge, Jr., Philadelphia, Pa. All of these papers are in the present volume.

C. R. Baird & Co., Philadelphia, Pa. (letter to Mr. W. R. Webster): Referring to your inquiry as to the requests of our customers for analyses of pig-iron, we beg to say that until the past few years very little attention was paid to the chemical constituents, and founders generally believed that it was necessary to use No. 1 X Foundry-iron for the manufacture of small castings, where softness and fluidity were essential. The quality and fitness of iron for this purpose was judged by the appearance of the fracture, and very large crystals were especially prized. An open-grained, soft iron covered their requirements. In larger machinery-castings, No. 2 X Foundry was mixed with No. 1 X, or, in many cases, used alone. It was classified as "somewhat closer and harder than No. 1 X," and was not as valuable as the former, because the crystals were

smaller. The next grade in price was No. 2 Plain Foundry, which was still closer in grain, and was used only for the heaviest and cheapest class of castings, or as a filler in connection with the higher grades. The light-colored, weak, high-silicon iron, made occasionally by every furnace, was regarded as poor stuff and utterly worthless for castings requiring strength and good finishing qualities. The founder depended absolutely upon the size of the crystals and the color of the fracture, and, if the iron looked all right, would, with no other guide, boldly attempt the manufacture of any kind of castings, even for the most difficult and expensive machinery, where defective castings might cause enormous loss. That such a state of affairs could have existed for years seems almost incredible, but no one will deny that such was the case. Recently, the greater knowledge of the effect of the chemical constituents of iron has been of immense advantage to founders; and this subject is now regarded as of the utmost importance. The influence of silicon is now commonly understood; but although other elements are entitled to almost equal recognition, they rarely receive it. It has been claimed that the appearance of the fracture, together with a knowledge of the amount of silicon present, is all that the founder requires to produce good results. This is, of course, better than the old method, where even the amount of silicon was unknown; but we find that the most progressive founders recognize the importance of the other elements, and request complete analyses. The *consensus* of opinion of these customers is as follows:

Elements.	Proportions Required.				
	I.	II.	III.	IV.	V.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Graphitic carbon	3.20	3.30	2.90	3.00	3.00
Combined "	0.30	0.20	0.40	0.30	0.30
Silicon	2.75	3.00	2.40	2.60	2.50
Phosphorus	0.60	0.80	0.60	0.80	0.90
Manganese	0.60	0.50	0.60	0.50	0.30
Sulphur	0.015	0.01	0.02	0.015	0.02

I.—No. 1 X Foundry for the manufacture of pulleys, small machinery-castings, etc.

II.—No. 1 X Foundry for carrying harder pig-iron or scrap, and for making choice light hardware-castings, etc.

III.—No. 2 X Foundry, for heavy machinery-castings, etc.

IV.—No. 2 X Foundry, for carrying harder iron, and for light machinery-castings, stove-plate, etc.

V.—No 2 Plain Foundry. There is a wide variation in the specification for this grade, which is more frequently sold without guaranty than any other. Many furnaces throw indiscriminately into the class of No. 2 Plain all iron that by fracture will not grade as No. 1 X or No. 2 X, and is too open and soft for forge-purposes. By these furnace-men and by founders in general, it is regarded as an inferior iron, to be used with caution; consequently the price is low, and it is often possible to obtain very valuable iron classified as No. 2 Plain; but it is probable that this grade will vary largely, unless graded by analysis. The largest consumer of No. 2 Plain in this region calls for the analysis shown in column V. above, and is able to meet his requirements at a low cost; but this is at the expense of other consumers, who do not specify analysis, and consequently get lower carbon and lower silicon, which is only the average run of No. 2 Plain iron. You, of course, realize the necessity of great variation in the specifications of the different elements in iron for widely different purposes. Where low-grade pig-iron or scrap can be secured at a low price, many use a large percentage of the same in connection with iron containing as high as 12 per cent. of silicon, with most satisfactory results. Where great fluidity is especially required, regardless of strength, iron very high in phosphorus is called for, etc. Owing to the fact that we handle the output of twenty-six furnaces, which produce all descriptions of pig-iron, and that we are very frequently obliged to furnish complete mixtures for various classes of castings, we have recently established our private laboratory, in charge of a competent chemist, and now make mixtures, and combine these widely different irons entirely by analysis. We gladly furnish, free of charge, a complete guaranteed analysis of not only our own brands, but of any pig-iron, castings or coke that founders may send us. We find this method by far the most accurate and satisfactory way of ascertaining the value of the different grades, and we think the subject is worthy of the most careful attention of all melters

of iron. In this connection, we beg to say that if we can be of any assistance to them it will give us great pleasure to place our laboratory at their disposal.

EDWARD K. LANDIS, Philadelphia, Pa.: In connection with the discussion of the physics of cast-iron, the following abstract of the results of investigation of the influence of the different elements, taken chiefly from the latest edition of Prof. Ledebur's treatise, may be of interest. This edition appeared in 1895, and doubtless many members do not yet possess it.

Carbon.—The affinity of iron for carbon is strong; hence all commercial iron contains carbon, though in some varieties of malleable iron the carbon is less than 0.1 per cent. When ignited with coal or carbonaceous solids, iron will absorb carbon to a degree dependent upon the temperature. Many hydrocarbons, as well as cyanogen and the cyanides, likewise yield carbon to iron when heated. Carbonic oxide, however, does not behave in this way, although, at low temperatures, carbon may be deposited from carbonic oxide upon iron which still contains iron oxides. In this case the carbon does not immediately unite with the iron. Notwithstanding the affinity above mentioned, the amount of carbon which iron will absorb is limited, not exceeding 4.6 per cent. (generally not more than 4 per cent.) of its weight. Manganese increases the absorption of carbon; silicon and sulphur decrease it, as does also phosphorus, but not so much as the others. Iron containing silicon or sulphur never contains as much carbon as the maximum for the kind of iron. These elements replace carbon nearly in the ratio of their atomic weights: one part silicon replaces $\frac{3}{7}$ part of carbon; one part of sulphur replaces $\frac{3}{8}$ part of carbon. Alloys of iron containing 10 to 20 per cent. of manganese may, in the absence of unfavorable ingredients, carry 5 per cent. of carbon, or somewhat more; with 35 per cent. of manganese the carbon may be 5.5 per cent.; with 50 per cent. of manganese, 6 per cent.; with 65 per cent. of manganese, 6.5 per cent.; with 80 per cent. of manganese, 7 per cent.; with 90 per cent. of manganese, 7.5 per cent. of carbon (and hence only about 2.5 per cent. of iron). On the other hand, iron with 2 per cent. of silicon and no manganese seldom contains more than 3.8 per cent. of carbon; and the carbon in ferro-manganese containing

50 per cent. of manganese and 2 per cent. of silicon does not much exceed 5 per cent. In a pure iron with 4 per cent. of carbon a small amount of silicon separates graphite. If only 1 per cent. of carbon is present, several per cent. of silicon may be present before the graphite can be observed. Manganese and sulphur prevent graphite formation. The more manganese or sulphur present the greater the amount of silicon required to produce graphite. The effect of the total percentage of carbon in iron is to lower the melting-point.

Silicon lowers the saturation-point for carbon and causes graphite separation. It also lowers the melting-point, but less than carbon.

Manganese increases the saturation-point for carbon, but prevents graphite-formation. Manganese and silicon have opposite effects on graphite. Manganese increases hardness and brittleness by increasing combined carbon. In large amounts it raises the melting-point. Manganiferous pig-iron has a tendency to absorb large quantities of gases, and thus produce blow-holes in the casting.

Sulphur reduces the total carbon. In large amounts it tends to produce low-carbon white iron. A few tenths of 1 per cent. of sulphur show this influence. In reducing the saturation-point for carbon, sulphur acts like silicon, but has an opposite effect to silicon in regard to graphite. It lowers the melting-point.

Phosphorus.—Iron phosphides without carbon are generally refractory; but high-carbon irons are more fusible. Phosphorus increases fluidity and hardness, but not as much as carbon. It lowers the melting-point. As silicon tends to form graphite, a high-silicon iron may contain more phosphorus than a low-silicon, low-graphite iron.

Moissan* has shown that the amount of carbon in pig-iron rises with the temperature at which the iron is formed. Ferromanganese and spiegel contain more carbon than pig-iron, and require a high temperature for their formation. It seems difficult to determine whether the high-carbon is produced by the presence of manganese *per se*, or by the high temperature required. Moissan found the graphite in a gray pig-iron to contain from 80 to 85 per cent. of carbon and 1.30 per cent. of

* *Comptes Rendus*, vol. 119, p. 1245, December, 1894.

ash, while graphite produced by adding silicon contained 98.82 per cent. of carbon and 0.85 per cent. of ash. The larger amount of ash in the first case might have been caused by the presence of other elements; and if this is the case it would explain the difference between iron naturally gray and iron made gray by the addition of silicon.

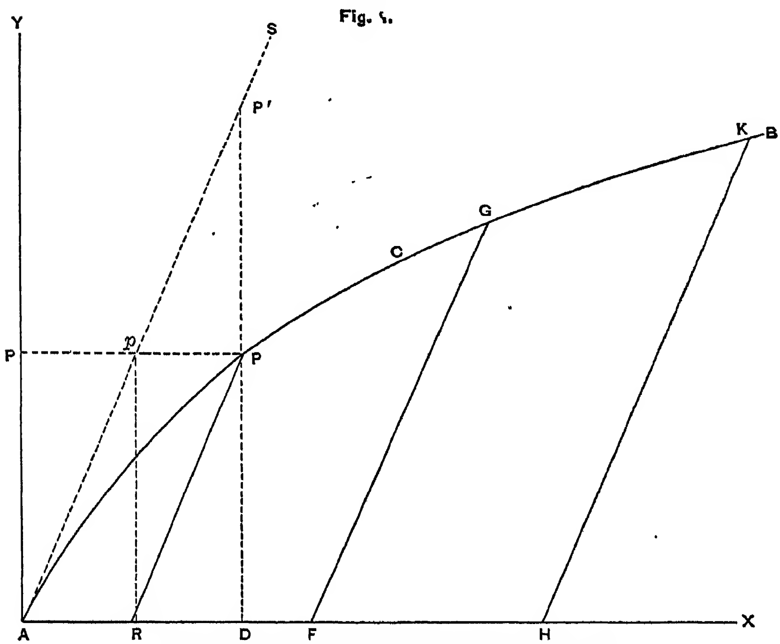
Moissan says also that the higher the temperature at which graphite is formed the purer the graphite and the more refractory towards oxidizing agents; also that the small amount of hydrogen present diminishes with the purity. Hitherto it seems to have been taken for granted that all graphites were alike; but Moissan's researches show that this is not the case.

PROF. R. C. CARPENTER, Ithaca, N. Y. (communicated by Mr. William Kent): *Modulus of Elasticity in Transverse Tests.*—By a study of the automatic diagrams obtained in the use of Keep's testing-machine about three years ago, the writer's attention was called to the following peculiar action of cast-iron after being strained. From the diagrams, the writer was led to believe that after cast-iron had been strained by the application of a load of any magnitude whatever, it received a certain set or deformation, but if that load were removed and a second one applied which did not exceed the first, the cast-iron would behave, within the limits to which it was first strained, like a perfectly elastic material. At my request, this property was investigated for larger specimens by Mr. C. E. Houghton, who confirmed in nearly every particular the conclusions drawn from a study of the diagrams from the Keep machine.

This peculiar action is best exhibited by a diagram, Fig. 1, in which distances measured parallel to the Y-axis denote stress applied to rupture the material, and distances parallel to the X-axis denote the corresponding deflection produced. The stress-and-strain diagram is represented by a curve, A C B, no portion of which is a straight line. Had the material been perfectly elastic, the deflection would have been proportional to the applied load, and the diagram would have been a straight line. Thus, for instance, if the deflection had been maintained at the same rate as in the initial application of load, the stress-and-strain diagram would have been represented by a straight line, tangent to the beginning of the curve and represented on the diagram by A S. Had the material been of the same stiff-

ness as represented by that line, we should have had a deflection, A D, with an application of a load, P′ D; but, because of the lack of elasticity, we actually obtain that deflection with the application of a smaller load, P D. If now the entire load be removed from the specimen, it will not regain its original form, but will be found to have a permanent set, A R. If a load be applied intermediate in amount between nothing and the original load, P D, the deflection will be found to be almost directly proportional to the load within these limits; that is, the material is elastic for any subsequent loads less than P D, and the strain-diagram becomes P R. A series of investigations proved that this phenomenon held good nearly to the breaking-point of the material; that in every case the material seemed to be rendered very nearly perfectly elastic by an application of a load; and that the elastic limit varies in amount proportional to the magnitude of the load.

FIG. 1.



Diagrams Showing Strains and Sets of Cast-Iron.

We may call the straight lines, P R, G F and K H, elasticity-

lines, and we find that these lines are not exactly parallel, but make a slightly increasing angle of inclination for increasing loads. The tangent of the angle of inclination of these lines would be the modulus of elasticity within the given limits of loading, which we would find from this investigation to decrease slightly with increase of load.

These elasticity-lines are, however, very nearly parallel and (until the load exceeds half the breaking-load) nearly straight; and they seem to the writer to represent very fairly the modulus of elasticity of the material after being strained.

Asa W. Whitney, Philadelphia, Pa.: By the use of Mr. Webster's method of noting the chemical and physical character of each test-piece upon a separate card, the following almost identical compositions have been picked out from a number of wheel-mixture tests, written up for study. It is not remarkable that the transverse strength and the contraction should agree so closely in different casts. That, of course, frequently occurs with very considerable difference of chemical analysis, compositions considerably different being counted upon to give nearly equal physical character in my system for practical iron-mixing by analysis only. But it first appears to be worth while to show that, under the same conditions, the same composition may be expected to give the same physical character.

The individual irons charged in the mixtures from which these tests were cast were not alike in grade, kind or chemical analysis; but the calculated chemical percentage-compositions of the charge on the given dates were identical, in the case of carbon and silicon, to the second decimal inclusive; in the case of manganese to the first decimal inclusive; and in that of sulphur to about the same degree. The details of the charge-mixtures are not given, as it may not be advisable, for commercial reasons, to expose the relations of these details to the resultant castings.

The conditions of melting were almost identical, as were those of moulding, pouring and cooling. The gray bars are hexagonal, 4 square inches in sectional area and 12 inches long between supports, and the chilled bars rectangular, 4 square inches in sectional area and 12 inches between supports. Gray bars are totally gray, and chilled bars are totally white (chilled through). They are the same shapes referred to in my contri-

bution to this discussion for the Florida Meeting last March. They are cast very close to size, but not corrected for any small variations or roughness of surface. Appreciable variations or flaws would have been noted. The figure for resilience is only approximately correct, partly because an average weight of bar is assumed in the formula,

$$\frac{\text{stress} \times \text{deflection}}{2 \times \text{pounds per bar}} = \text{inch-pounds per pound of metal.}$$

The chilled bars may not have been poured from the same hand-ladle of iron, but were poured within a minute of the gray-bar in each case, and before another tap of iron into the large mixing-ladle. These bars are always poured as hot as practicable.

Comparative Tests from Different Mixtures I. and II.

(Being A. Whitney and Sons' Regular Wheel-Mixture of Nearly Identical Chemical Composition. Middle of Melt on Each Date.)

Date of Casting: I., September 12, 1895; II., October 1, 1895.

Analysis. Per cent.			Physical Tests.				
				Gray Bar.		Chilled Bar.	
	I.	II.		I.	II.	I.	II.
C	3.799	3.799	Stress, pounds	26,000	25,500	14,000	16,500
Si	0.606	0.602	Modulus of rupture, pounds	57,700	56,590	42,000	49,500
Mn	0.485	0.483	Resilience, inch-pounds per pound	109.8	80.9	10.8	14.1
P	0.328	0.325	Contraction, inches in 12 inches	0.14	0.14	0.24	0.22
S	0.106	0.088					

Average tape* of 33-inch wheels (dependent on the contraction of the iron), inches: I., 117; II., 117.

Depth of chill in inches: I., $\frac{1}{2}$ to $\frac{7}{8}$; II., $\frac{1}{2}$ to $\frac{3}{4}$.

The unchilled bars were, in both cases, good open, gray iron.

On both the dates given in the above table, wheels were made from the respective mixtures for the Pennsylvania and the Philadelphia and Reading Railroads, and passed inspection. The specifications of the Pennsylvania Railroad Company are appended to this communication, as an interesting and valuable record of practice. They were issued in June, 1891, replacing

* That is, measurement by steel tape around circumference.

earlier specifications of the same company, issued in 1888, and representing the results of later study on the part of the company's experts, and added experience in the use of cast-iron wheels over thousands of miles of track. The specifications of the P. & R. R.R. are practically equivalent to those of the Pa. R.R.

The above instance indicates the practicability of obtaining castings of the same qualities from different mixtures, by combining the constituent pig-irons in such proportions as to secure from the total mixture approximately the same chemical composition. This simple synthetic method is independent of the question, what may be the nature and degree of the influence of each separate element, and, taken by itself, the above example throws no light on that question. It is merely one, out of innumerable instances which might be given, to prove the direct and controlling relation between the chemical composition and the physical qualities of cast-iron. This question is, after all, the fundamental one, even to science; for if a given chemical composition does not yield always, under similar treatment, the same physical qualities, it is comparatively useless, as well as impracticable, to attempt to estimate the relative and separate effects of its several constituents. To the practical founder, the proof that a composition once shown to produce the desired result may always be depended upon to do so, no matter what original irons may have been fused to make the mixture having that composition, is of the highest practical importance, as delivering him from traditional superstitions concerning the mysterious qualities of certain special ores and brands of pig.

As implied in the first paragraph of these remarks, a further step in the application of scientific principles to the founder's business lies in the discovery that practical similarity in the desired physical qualities may be secured from different compositions. This bears upon the difficult problem of determining not only the qualitative, but also the quantitative effect of each ingredient, and (most difficult of all) the modification of such effects by the presence and amount of other ingredients. I am not prepared to present at this time any digest of the results of practice likely to throw light upon this complicated question. But I will offer the following instance, indi-

cating that minute changes in chemical composition may be accompanied with decided differences in physical appearance. The case is but one of a number which have come under my notice, all of which, though differing considerably in details, indicate sensitiveness to small variations in composition.

Analyses of Different Parts of the Same Pig, Per Cent.

	C.	Si.	Mn.	P.	S.
I., .	4.339	0.193	1.027	0.253	0.02 or less.
II., .	4.255	0.122	1.016	0.273	0.02 "

I. The gray part of the pig; dark, open and soft II. A spot, occupying two-thirds of the sectional area of the center of the pig, and the bottom; white, but not very hard.

This is far less variation than is often found in different parts of a gray pig or casting. Yet, notwithstanding this apparent unreliability, such irons may be safely used for any purpose if properly mixed with suitable other irons.

As stated in my contribution of a year ago to this discussion, the control by chemistry of all our regular and special mixtures since January, 1892, is due to an increasing knowledge of what I may term the qualitative value of various compositions by means of a general key, which when closely worked out will doubtless explain quantitatively even the most puzzling cases. By practice and continual study with this key, applied to the comparison of analyses since 1884, a general practical judgment of the quantitative value also of various compositions, has been obtained by the writer, enabling him to use the elements and their apparent combinations in cast-iron as units with practical certainty and economy, and over a far greater range of material and product than he could attain by considering each particular grade and appearance of iron as a unit.

Mr. William R. Webster was recently engaged to tabulate this work for us, whereupon our Mr. C. F. Fisher, my assistant since 1891, became further interested in the work, and by Mr. Webster's method of writing up records and his own study thereon, he has made a good start upon a table to show the transverse strength of our regular hexagon bar when the complete analysis is given.

By the application of these tables and, in many cases, in a more direct manner, from study and from some work kept in hand by the writer as opportunity has offered since 1889 on

grade in general, strength and chill, the results of this system of iron-mixing will eventually be even more exact through a range of character covering all classes of castings, the differing conditions of manipulation being properly taken into account.

In evidence of continued success in the preparation of casting-mixtures for special strength, coupled with excellent tooling-quality and capacity for fine finish, I may instance four special mixtures, made up at our works since January 1, 1896, by calculation from analyses only of stock on hand and of various castings of approximately the desired character, to secure high strength and proper softness. Two of them, warranted to have a tensile strength of at least 30,000 pounds per square inch, gave respectively 37,400 and 37,000 pounds, in test-pieces turned from castings 2 inches in diameter and 9 inches long. The other two were expected to show a tensile strength of over 30,000, and actually exhibited 35,840 and 35,750 pounds per square inch, on similar test-pieces, turned from the center of half the regular hexagonal bar of 4 square inches sectional area.

A few days ago, I learned that the bottle-mould made by us, and referred to in my contribution to this discussion at the Florida meeting, is not yet worn out, after twenty-two months' service; from two to three months being the service obtained from other compositions.

Specifications of the Pa. R. R. Co. for 33-inch Cast-Iron Wheels.

Design.—The design of wheels must be such that they will be in accordance with the measurements shown in the drawings accompanying these specifications, and also such that the wheels made from them shall weigh not less than 540 pounds nor more than 575 pounds each. The use of brackets in ring-core surrounding the hub will not be permitted.

Marks.—Each wheel must have plainly cast upon it the name of maker, and on the inside double plate the date of casting and a serial number. No wheel bearing a duplicate number or upon which the number is not distinct will be accepted, nor shall one or more figures contained in the number be altered or removed to avoid this clause. Numbers made vacant by a rejection or otherwise must not be filled.

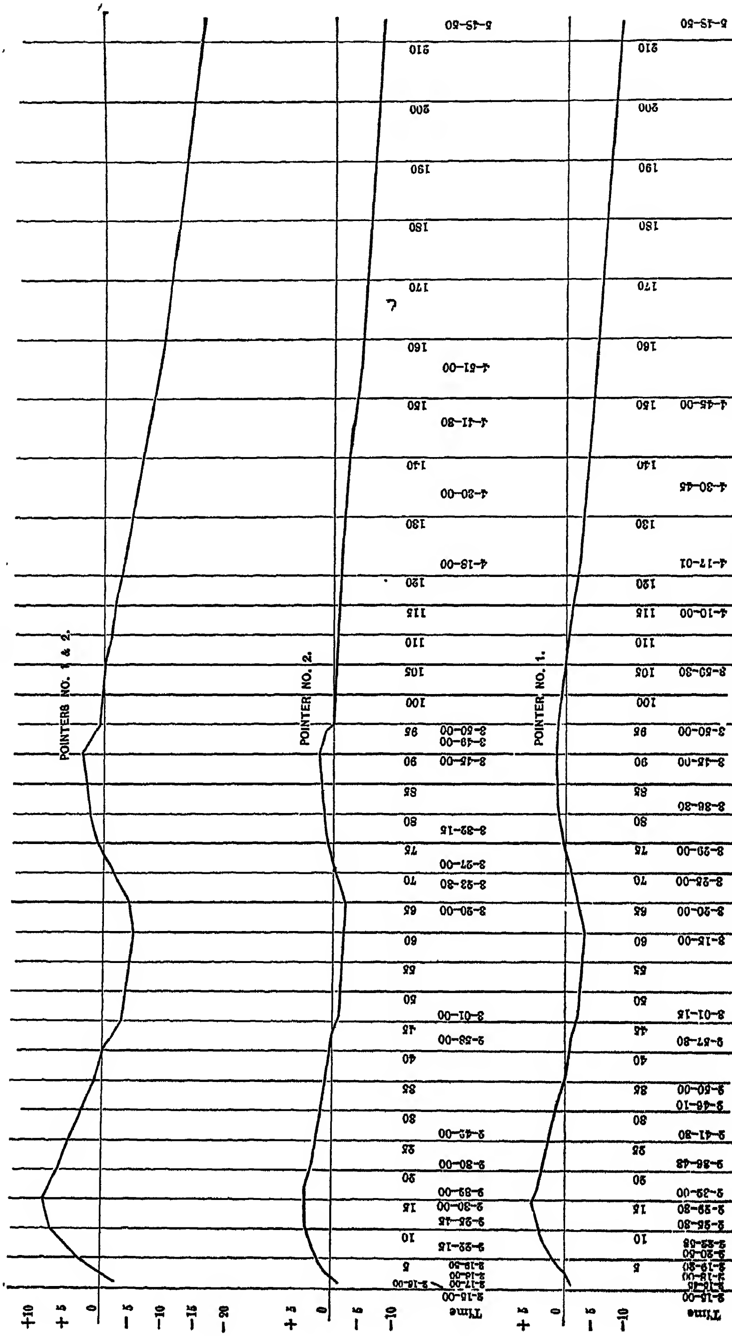
Chills.—All chills must have the same inside profile, as per accompanying drawing, their inside diameter measured at throat must be the same, and they must be truly circular. Chills in which checks or fire-cracks are more than $\frac{1}{32}$ inch wide and over 2 inches long, so as to produce undulations in the contour of the tread or flange of wheels cast in them, will not be considered as suitable for their purpose, and wheels made from them will be rejected.

Inspection.—All wheels offered for inspection must be measured with a Pa. R. R.

standard tape-measure, and must have the shrinkage-number stencilled in plain figures on the inside double plate of the wheel. When ready for inspection, the wheels should be arranged in rows according to shrinkage-numbers, all wheels of the same date being grouped together. All wheels must, during inspection, receive three heavy blows with a 6-pound sledge at as many different points under the flange between the brackets without cracking the flange or brackets; and this must be rigidly enforced, especially with wheels showing high shrinkage. No wheel will be accepted, the circumference of which is more than $1\frac{5}{8}$ inches, or less than $\frac{7}{8}$ inch, smaller than that of the chill in which it is cast. If wheels are presented which have been made in contracting-chill, the limit of shrinkage will be 2 inches. Each wheel must be so nearly circular that when a true metallic ring is placed on the tread and bears somewhere on the cone, it shall at no part of the circumference stand more than $\frac{1}{16}$-inch away from the tread. Body of wheel must be smooth and free from slag and blow-holes, and hubs must be solid and free from drawing. Tread and throat of wheel must be smooth, free from deep and irregular wrinkles, slag, sand-wash- and chill-cracks, and throat must be practically free from sweat. Wheels tested must show soft, clean gray iron, free from defects, such as holes containing slag or dirt more than $\frac{1}{4}$-inch in diameter, or clusters of such holes; honey-combing of iron in the hub; white iron in the plates or hub; or clear white iron around anchor or chaplets at a greater distance than $\frac{1}{2}$-inch in any direction. The depth of clear white iron must not exceed $\frac{7}{8}$-inch at throat and 1 inch at middle of tread, nor be less than $\frac{3}{8}$-inch at throat and $\frac{5}{8}$-inch at middle of tread. The blending of the white iron with the gray iron behind it must be without any distinct line of demarcation, and the iron must not have a mottled appearance in any part of the wheel, at a greater distance than $1\frac{5}{8}$ inches from the tread or throat. Wheels inspected and tested, and failing to meet these requirements in any particular will be sufficient cause for the rejection of all wheels which they represent, in like manner as those failing to stand the test for strength.

Test.—For each fifty (or fraction thereof) wheels that pass inspection and are ready for shipment, one wheel shall be selected by this Company's inspector, and subjected to the following test: The wheel shall be placed, flange downward on an anvil block weighing not less than 1700 pounds, set on rubble masonry 2 feet deep, and having three supports not more than 5 inches wide for the flange of the wheel to rest upon; it shall be struck centrally upon the hub by a weight of 140 pounds falling from a height of 12 feet. Should the wheel break in two or more pieces after eight blows or less, all the wheels represented by this test-wheel will be rejected; in this case, the inspector may, if he so elect, consider the wheel tested to represent all of the wheels of the same shrinkage-number and date of casting, and will be at liberty to test in the same manner one extra wheel of each and every other shrinkage-number of the wheels cast on the same day. Should the wheel tested stand eight blows without breaking in two or more pieces, the wheels represented by it will be accepted, and may be shipped; provided, however, that the requirements in regard to depth and uniformity of chill are met; but in all cases shipments will be subject to return of such wheels as are found, upon boring and mounting, to be below these specifications in any respect. It shall always be the privilege of this Company's inspector to test one wheel from each day's foundry-work. A machine for making these tests is shown in the accompanying drawing, and manufacturers furnishing wheels to this Company will be required to provide one of these machines and furnish wheels for test as well as such facilities to this Company's inspector as will enable him to inspect and test wheels promptly.

FIG. 2.

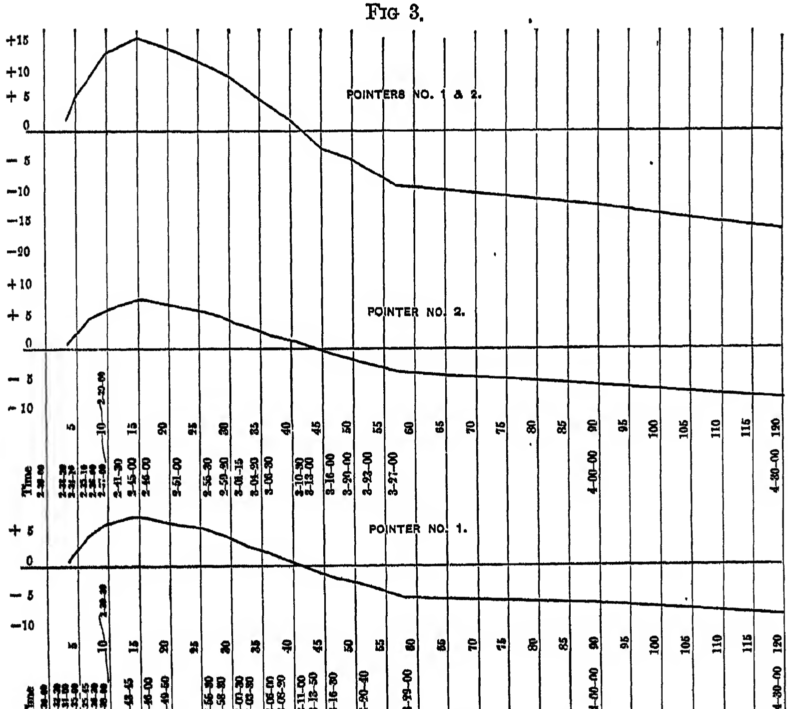


HINKLE NO. 3, EXPANSION AND SHRINKAGE TEST, MAY 24, 1889

NOTE.—After receiving 10 blows of the Pa. R.R. drop above described, the wheels are rolled to another drop of 700 pounds, falling 10 feet, and require 1 to 5 (usually 2 or 3) blows of this to break them into two pieces. Occasionally they are broken at the Pa. R.R. drop, requiring usually from 15 to 40 (frequently as much as 50, and in some cases 75 to 100) blows of that drop. One wheel endured 425 blows of the Pa. R.R. drop (though the first crack appeared at the 20th blow); and 3 blows of a 500-pound drop falling 10 feet were then required to break it into two pieces.

WILLIAM R. WEBSTER, Philadelphia, Pa.: In 1889 Mr. John R. Whitney, of Messrs. A. Whitney & Sons, Philadelphia, made a number of expansion and shrinkage tests upon molten and solidifying cast-iron. He referred to this investigation in the discussion of an article entitled "Behavior of Iron in Solidifying," in the *National Car and Locomotive Builder*, May, 1889.

FIG. 3.

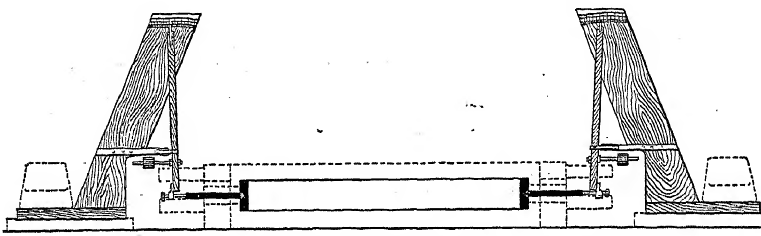


WOODSTOCK NO 3, EXPANSION AND SHRINKAGE TEST, MAY 27, 1889.

Messrs. A. Whitney & Sons have furnished me with the full records of this investigation, and I have plotted, in Figs. 2 and 3, the results of two of the tests in which all the conditions were the same, except the irons used. In the first case, Hinkle No. 3 (Lake Superior charcoal) was used, and in the second Woodstock No. 3 (Alabama charcoal).

Bars $2\frac{5}{8}$ inches wide by $3\frac{13}{16}$ inches deep and 3 feet $1\frac{1}{2}$ inches

FIG. 4.

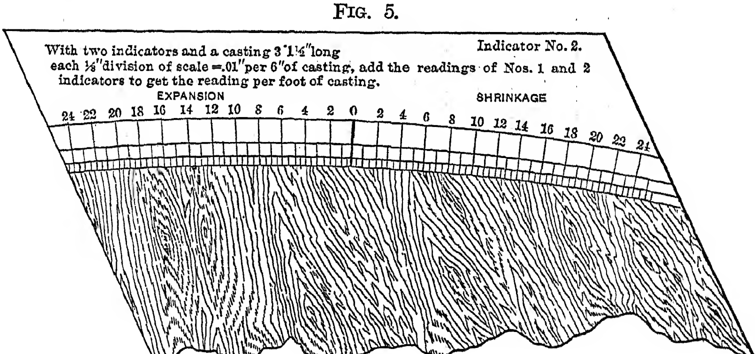


Whitney's Apparatus for Testing Expansion and Shrinkage.
Casting, 3 feet $1\frac{1}{2}$ inches long, $2\frac{5}{8}$ inches wide, and $3\frac{13}{16}$ inches deep.

long were cast in a closed flask. The cope was not removed in either case until the next morning, when the castings were too hot to handle with bare hands.

Figs. 4 and 5 show the apparatus used and general arrange-

FIG. 5.



Graduated Scale. One-half of Natural Size.

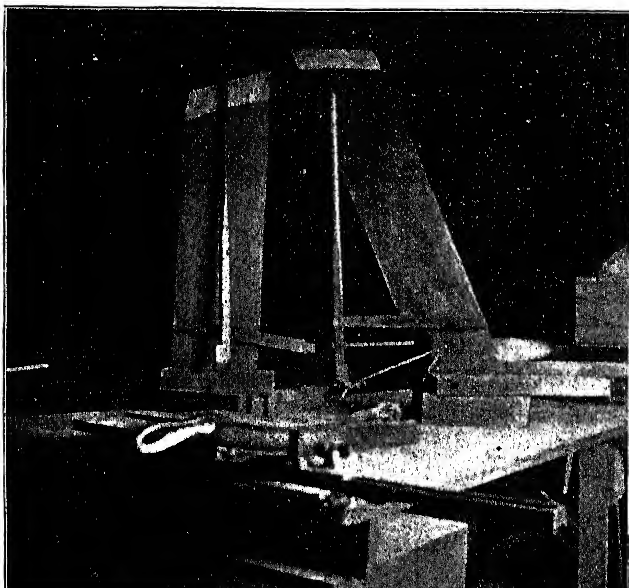
ments of pointers, etc. Pieces of electric-light carbon were used in each end of the casting. When so arranged, each $\frac{1}{8}$-inch division of the scale of each indicator represents a motion of 0.01 inch per 6 inches of casting. To obtain the expansion or shrinkage per foot at any given time, add the read-

ings of the two indicators at that time together. The sign + indicates expansion and the sign — indicates shrinkage.

The electrical attachment and galvanometer were used in order to check the contact of iron and carbon. The contact was not broken in any case.

In Figs. 2 and 3 the expansions are plotted above the horizontal lines and the contractions below the horizontal lines.

Fig. 6.



Remains of Whitney Apparatus of 1889.

The figures given in the columns at the left side represent thirty-seconds of an inch on the scale of the indicator. The time of each observation is given on the horizontal line. This is given for each pointer, and at the top of the diagram for both added together.

In the case of the Hinkle iron you will notice two expansions, while in the Woodstock iron there is only one expansion. In all the 17 tests made, the Hinkle iron was the only one that showed this second expansion. All the others showed the first expansion, similar to the Woodstock iron.

Fig. 6 is from a photograph, showing the remains of the ap-

paratus for testing expansion and shrinkage used at the Whitney works in 1889. Both pointers were originally like the one on the right in the figure. A weight of about 5 pounds was hung on the projecting iron rod.

Mr. Whitney's attention having been called to the experiments of Mr. Wrightson and of Mr. Charles Markham, given in their papers before the Iron and Steel Institute in 1879, 1880 and 1881, he stopped his investigation on finding that so much had been done by others, and that it was necessary to measure the width and thickness before any reliable results could be obtained. I call particular attention to this in view of the recent investigations in this same line of work, which involved the same defect.

Mr. Markham stated that Sir Lowthian Bell, in his experiments (on 6-inch cylinders, 72 inches long) found that cast-iron expanded for an hour, whereas he (Mr. Markham) showed in his experiment (on 4-inch cylinders, 9 feet 4 inches long, and rectangular pieces 4 by 5 inches and 30 feet long) that no expansion took place after the iron was run.

Mr. Wrightson called attention to the important fact that Mr. Markham had not taken any measurements of width and thickness, and therefore could not show either increase or decrease in volume.

As this matter now stands, we need more light on it before drawing any general conclusions from a few observations.

William Kent, Passaic, N. J.: Mr. Outerbridge's paper, it seems to me, is one of the most remarkable ever presented to the Institute, bringing forward, as it does, new ideas. Mr. Outerbridge tapped a bar 3,000 times, and he says that the tapping strengthened it. If he had kept on tapping it for 3,000,000 times he might have disintegrated it and broken it up. It would be well to determine whether the law of increase of strength with molecular annealing cannot be carried too far. That seems to be what happens if we hammer wrought-iron long enough. Cast-iron also has generally been believed to have a certain life, and if you hammer it long enough it will break. I have known of a cast-iron shaft that stood up to its work in a rolling-mill for 25 years; but then it broke. At one of the meetings of the Institute we saw a specimen of pig-iron which was broken by 400 blows of a sledge-hammer.

Certainly, the blows may have strengthened it at first. So we do not know yet all about the effects of vibration on cast-iron.

The paper on "Contraction and Expansion," and Mr. Whitney's and Mr. Webster's experiments, are highly interesting, and bring us back to where we were in 1879–80, according to papers published at that time in the *Proceedings* of the Iron and Steel Institute of Great Britain. At that time the whole question of the contraction and expansion of iron due to solidification was left in a state of uncertainty. We are perhaps a little farther along now, but not much. Mr. Keep, of Detroit, presented to the American Society of Mechanical Engineers last year some diagrams showing a peculiar double expansion. After the iron began to solidify it expanded, then it contracted, and then it expanded again. Mr. Keep announced that as a new discovery of the double expansion of iron, or, as he called it, the second expansion. Now, Mr. Webster presents some experiments made by Mr. Whitney, in Philadelphia, several years ago, and this double line was then found only in one case out of seventeen. So it is interesting to know under what conditions that double expansion takes place. Mr. Keep's experiments, and also Mr. Whitney's, seem open to the criticism that was made by Mr. Thomas Wrightson, in 1880, on Mr. Markham's paper, read before the Iron and Steel Institute. Mr. Wrightson proved that cast-iron expanded during solidification, and Mr. Markham proved the exact contrary. Then Mr. Wrightson made the criticism of Mr. Markham's experiments that the measurements were taken of one dimension only—that is, the dimension in length. He said the proper thing to do was to measure the volume or the dimensions in three directions. Mr. Wrightson himself did that, and apparently has proved that iron does expand during solidification or immediately afterward, and contracts regularly after that. There is no evidence in Mr. Wrightson's experiment, nor in those of Sir Lowthian Bell, made shortly after, which shows any signs of this double expansion, which was recently discovered by Mr. Keep and discovered by Mr. Whitney several years before.

Mr. West has given us something about the proper shape of test-pieces for cast-iron. Mr. Keep, in his papers before this and other societies, has argued the advisability of a square test-

piece. I think it would be well to appoint a committee of two, consisting of Mr. Keep and Mr. West, to reconcile their ideas and then present a new paper. I would suggest to these gentlemen who are now discussing what is the proper shape for a test-piece for cast-iron a new idea which they may think over. It is well known that the strength of cast-iron is not only dependent upon the analysis of the iron and upon the heat at which it is cast, but upon the shape and size of the test-piece; that is, if you have iron of a given analysis and make a test-piece of a bar ½-inch square and another of a bar 2 inches square you will find entirely different results. The iron will differ both in strength and in ductility, and that difference is not a constant difference, but it depends upon the chemical constitution of the iron. So the idea I wish to throw out is, that one standard size of test-piece of cast-iron is not enough, and I suggest that three or four different-sized test-pieces be used. I would also indicate a way of plotting the results for the purpose of studying them. Suppose we find that the round shape is the better and we select a ½-inch, a 1½-inch, and a 2-inch as the best sizes for comparison to be tested. Suppose we then test four different qualities of iron, containing respectively 0.5, 1, 2 and 3 per cent. silicon, four test-pieces of each, and obtain the transverse strength. If we divide the results obtained in each test by the product of the breadth by the square of the depth, in inches, that is by bd^2, if the test-pieces are of rectangular section, or by the cube of the diameter, if they are of round section, the figures will be reduced to a standard of comparison, if the pieces are of uniform length between supports. A better standard of comparison, perhaps, is the modulus of rupture, obtained from the usual formula $R = \frac{3}{2} \cdot \frac{Pl}{bd^2}$, in which P is the breaking-load in pounds, and l, b and d are respectively the length, breadth and depth in inches. Suppose that the several results we obtain, in terms of breaking-load divided by bd^2, are the figures in the following table:

Silicon. Per cent.	Diameter of Test-Piece. Inches. 0.5.	1.	1.5.	2.
0.5,	800	1100	1300	1400
1.0,	900	1050	1100	1100
2.0,	1100	1200	1150	1050
3.0,	1350	1300	1200	900

A similar table may be made of the deflections obtained in the tests. To reduce the deflections to a standard they should be multiplied by bd^3 and divided by l^3.

We now plot these figures and obtain the diagram given in Fig. 7. Each of the four curves in the diagram expresses, far more clearly than any test of a single size of test-piece can do, the nature of the iron tested. I offer this suggestion to those of our members who are engaged in testing cast-iron, with the confident expectation that the proposed method will lead to important discoveries as to the relation existing between the chemical constitution and the physical properties of cast metals.

FIG. 7.

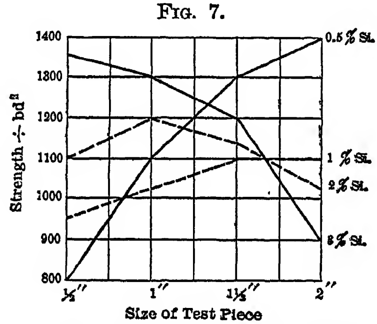


Diagram Showing Relations of Strength, Size, and Silicon-Contents of Iron Castings.

LEONARD WALDO, Bridgeport, Conn.: This question of physical tests of metals is one of fundamental importance, and is hedged about with difficulties. I think it was Professor Roberts who showed that if you take a cube of aluminum bronze, divide it off into smaller cubes by drawing lines in it, and make an analysis of each of the smaller cubes, you will find that the metal is homogeneous from end to end. It offers, therefore, an excellent opportunity for the testing of strains in metal, free from the question of segregation, because it is uniform throughout, and you are free from that difficult part of the problem which relates to the unequal distribution of its chemical constituents. Acting on that suggestion, which came from reading Professor Roberts's work, I prepared a series of bars running up to an ingot 4 inches square and made of aluminum bronze, and, while not expecting to take part in this discussion, I have

no accurate figures with me, broadly speaking, it was shown that the ultimate tensile strength was dependent upon the diameter of the section employed, and that the tensile strength diminished from the exterior of the bar to the interior of the bar. Our lamented Eckley B. Coxe was talking in the laboratory of the McGill University at the time of an inspection of the magnificent mechanical laboratory which he was looking over, and he turned to Professor Woodward and said: "What a capital thing it would be if you fellows in laboratories and schools should simply put up a bridge-truss and break it down and see what it meant." The truth underlying that comes home to every man who has to build a structural engineering pattern in metal. I cannot imagine that a test-bar is of any use except in a very general and qualitative sort of way for determining the actual strength of metal that is in the structure and that has been cast in the structure. The entire question of flow, cooling and chill enters, and, while you may produce your lead-pencil test-bars of the kind which will have tensile strength from 80,000 to 90,000 pounds per square inch, the moment that you take a large bar 2 inches square, or something which will approximate the size of the actual metal used in the construction, your tensile strength drops down with the same chemical constitution to 40,000 or 50,000 pounds, and you get very discouraging results. I took a round-rolled bar 1¾ inches in diameter and cut the bar in two lengths, each about 18 inches long. I then bored one through so as to cut out the entire center of the bar, and turned down the other so that I got corresponding areas—one the circular ring and the other the actual area of the turned-down bar—and I found that the difference in tensile strength amounted to about 15 per cent. between the inside of a bar of that size and the outside. I sent to the Watertown testing-machine, at Waltham, Mass., a series of large bars up to 4 inches in diameter, hoping to get the test there, but I found that their machine had been so badly used up by some strains of high pressure that they were unwilling to break the bars on the machine. The machine is commonly rated at 800,000 pounds, but the last experiment approximating three-quarters of that amount ended by shearing off all the nut-heads at the recoil of the machine, and I found my labor was in vain. So that, in the question of determining these strains,

much as we may resolve or desire, I think I am safe in saying that there is no machine in the country which would actually take a piece of structural metal of high quality and break it with accurate registration of results, and that our knowledge must remain more or less uncertain until that is done. It seems to me that what is wanted is not small tests in a small way, but tests of material as we actually use it in practice; but I see no immediate way of getting such satisfactory tests. I have talked some with the Ordnance people and those who have to build these large things, and I find that without any exception they are extremely shaky as to the strains which result in the interior of castings of large masses of metal from chill on the outside, the natural arching of the metal, and the molecular strains under which the metals exist.

George S. Morison, Chicago, Ill.: I should like to ask what would be considered a large strain to be put upon a testing-machine? There are in the country two large testing-machines, both of which I have had occasion to use, but they have not the accuracy of registration which the Waltham machine has. I have found that I could not depend on the Waltham machine, because it was always busy. There are two machines of much larger capacity: one of 1,200,000 pounds, and another of 2,000,000 pounds, both of which I have had occasion to use in testing structure steel. One is at Athens, Pa., and the other is at Phœnixville. They are both hydraulic machines working under comparatively low pressures; being hydraulic machines and registering their strains by hydraulic gauges, they are not as accurate as the Waltham machine; but for a number of years they have proved very much more serviceable. The Athens machine was built to meet requirements that I put into a specification; it was originally used for testing the eye-bars used in the building of the Omaha bridge. Since the Phœnix people built a machine which has a capacity of 2,000,000 pounds, I have had bars tested at Phœnixville which exceeded the capacity of the Athens machine.

A. E. Outerbridge, Jr., Philadelphia, Pa.: Since the publication of my paper, I have received from others, who have repeated the experiments, records of the gain in transverse strength exceeding the maximum reported by me. The largest

gain noted in my paper (on a 1-inch bar) was 525 pounds, or not quite 19 per cent. The most recent report, from a large establishment in Buffalo, shows in one case 780 pounds, or more than 25 per cent. Records from another establishment give more than 30 per cent. gain. In one instance, four bars were placed in a tumbling barrel, *without any other castings*, for 2½ hours. These showed gains ranging from 10.8 to 20 per cent., which surprised me, although it proved my statement that heavy blows were not required. The abnormal gains were in these cases on tests of strong iron. One bar not "rumbled" broke at 3060 pounds; the companion-bar, "rumbled," broke at 3840 pounds.

R. W. RAYMOND, New York City: On page 987 of Vol. xxv. in a foot-note to the contribution of Mr. A. W. Whitney, a calculation of the modulus of rupture of a hexagon bar of 4 square inches sectional area is given on the authority of Mr. C. F. Fisher. Since the publication of this part of the discussion, my attention has been called by Mr. Whitney to an error in this calculation. Namely, in the final equation, $3 \times 1.074 \div 1.406$ is 2.2918, instead of 2.22, as stated. But further investigation reveals other apparent errors. In the first place, according to the statement of the method given in the note, the element G is the distance between the center of gravity of the half-section and that of the whole section, and this distance is given for the section of 4 square inches hexagonal area as 0.593 inches, whereas it is in reality, as the following calculation shows, 0.477 inches.

The hexagon bar being conceived as lying upon one side, the neutral plane will pass through two opposite corners of the hexagon, and will contain the center of gravity (which is in this case the center of magnitude) of the whole section. The two half-sections, respectively above and below this plane, are symmetrical, and constitute truncated triangles. The center of magnitude of each of these figures is easily determined by Rule 4, on p. 82 of Rankine's *Rules and Tables*, for the determination of the center of magnitude of "a compound figure formed by subtraction." It will be found to be distant from the neutral plane by two-ninths of the minimum diameter of the hexagon. For a hexagon having 4 square inches of area, this minimum diameter (or the length of a perpendicular between two opposite sides) is 2.1492 inches. Consequently the element

G, or the distance between the center of gravity of the half-section and that of the whole section, is 0.4776 inches, instead of 0.593 inches, as given in the foot-note under consideration. By substituting this value in the calculation as given, we would get 0.91244 for the moment of inertia of the whole section, and 3.533 S for the modulus of rupture. But this result is certainly too high, as has been abundantly shown in practice. The soundness of the method followed is, however, more than doubtful. According to an independent calculation upon the usual method, the moment of inertia would be 1.28, and the modulus of rupture consequently 2.51 S This would be more nearly in accordance with the results of practice. It is to be remembered, however, that the modulus of rupture is an imaginary factor, involving unproved assumptions (such as the equality of the resistances to compression and extension) and only applicable for rough approximations. As the subject has only a mathematical interest for civil engineers, I will not enter here upon a detailed discussion of the theoretical methods of determining the moment of inertia or the modulus of rupture, but will only add that in Mr. Whitney's practice the hexagonal test-bars of wheel-iron frequently endure a stress of 26,000 pounds, and that the maximum modulus of rupture is about 70,000 pounds, which would be in the neighborhood of 2.5 S. In average practice I believe he assumes 2.25 S as the modulus of rupture.

In this connection I may call attention to a typographical error in the next following paragraph of the foot-note referred to (*Trans.*, xxv., 987), where square bars "2 by 3" inches in section should read "2 by 2" inches, the area being 4 square inches as for the hexagon bars. The modulus of rupture, 2.25 times the stress, is here correctly given.

David Townsend, Philadelphia, Pa.: I am exceedingly glad that at last a move has been made to throw some light on a subject of such universal interest. The manufacture of steel-products was created with the birth of a new industry, and was not therefore hampered by tradition. With the manufacture of steel came also the introduction of complete chemical and physical laboratories, by means of which the effect of various combinations could be recorded and studied. Cast-iron, on the other hand, has been treated with great conservatism on ac-

count of its age and history; and the light which, it is hoped, may be the beginning of a new era in that history, has been long in coming.

I had occasion, in 1878, to make for Mr. James Moore, of Philadelphia, a series of investigations, which had for their object the discovery of a proper chemical mixture of irons for making chilled rolls, and the discovery, if possible, of the cause of cracking in chilled castings. I made a complete analysis of 20 to 25 chilled rolls which had proved good in service. They were of all sizes, from 30 inches down to 10 inches in diameter.

The results were somewhat remarkable. The greatest differences in composition appeared in those rolls which were known to have been cast from the cupola. The air-furnace rolls had the same chemical composition within very narrow limits.

One particular point struck me as being rather curious; namely, that a certain percentage of sulphur seemed to be necessary to make a good chilled roll. The reason for this did not appear clear to me until Mr. West presented me to-day with corroborating evidence that the presence of sulphur in cast-iron, other elements being equal, produced an expansion at the instant of solidification, and, therefore, when a chilled skin was being formed on the roll, and that this prevented the formation of shrinkage-cracks.

Of the other elements, manganese and silicon have most influence on the chill; the former affecting its hardness and the latter, if present in sufficient quantity, preventing its formation entirely.

It was my invariable experience that low-sulphur irons would make chill-cracks, especially when the percentage of silicon was also low.

I am glad to have this corroboration from Mr. West, and I also desire to take this opportunity to thank him for his original and various contributions to the subject under discussion.

As to the physical properties of cast-iron, I quite agree with Mr. Kent that they differ for different sizes of test-bar; and it is therefore my hope that some standard, which will strike an average for all, will be adopted or recommended by those whose opinions will carry some weight. My own experience of twenty years has been altogether with charcoal-irons melted in an air-furnace; and I have never had success in recording

results with any but round bars, which were never less than 1 inch, and were usually 1¼ inches in diameter. The tendency of charcoal-irons to chill on the edges makes a square bar impossible as a means for making records.

We are at present making some castings under specifications requiring a tensile strength of 28,000 pounds. In sixteen different heats, composed of 5000 pounds each, a round turned test-piece of 1⅞ inches in diameter showed a variation of only 2000 pounds, the limits being 32,000 and 34,000 pounds.

I hope the good work will not cease until the ancient and honorable subject of cast-iron has been thoroughly investigated, and a practical solution has been found for all the difficulties which still beset the manipulators of this metal.

William R. Webster, Philadelphia, Pa.: I have personally witnessed a number of the experiments reported in Mr. Outerbridge's paper on "The Mobility of Molecules of Cast-Iron." The space which exists between the particles of cast-iron can be demonstrated by a very simple experiment, which was shown to me by Mr. Outerbridge, and also independently by an old founder, who said he had been familiar with it for eighteen years. It consists in brushing out the carbon from a surface of fracture of gray iron. Such a surface having the well-known gray and graphitic appearance can be made to look like hard white-iron by simple brushing with a wisp-broom or a brush of bristles. (Mr. Webster exhibited this experiment upon a piece of gray iron, covering half of the surface of fracture with paper, to protect it from the brush, and then brushing the exposed half. After the removal of the paper the iron looked as if it were half gray and half white. He also brushed a piece of iron having a chill on it, and after brushing, the whole fracture, soft iron and chill alike, was of one color, as if the chill had gone all the way through it. It was not possible to tell from the color where the chill stopped.)

A. E. Outerbridge, Jr., Philadelphia, Pa. (Communication to the Secretary, February, 1897): The *Iron Trade Review* of January 7, 1897, contains a paper entitled "Freaks of Foundry Iron," read December 28, 1896, before the Pittsburgh Foundrymen's Association, by Dr. A. B. Harrison, chemist of the Clinton Iron and Steel Company. Dr. Harrison had, apparently, read merely an extract made by a trade-journal from my paper

on "The Mobility of Molecules of Cast-Iron," presented at the Pittsburgh meeting of the Institute in February, 1896. He said:

"Having read in one of the trade journals of the effect of 'tumbling' test-bars in the barrel, we decided it might be interesting and useful to gather some data ourselves. So we have had several such tests made, taking one of the tensile-bars for usual test, putting the other bar (always casting these in pairs on one gate, as mentioned before) in the barrel, tumbling them once a day."

The results of his independent repetition of my experiments are interesting. The maximum gain in transverse strength, due to rumbling the bars, as recorded in my paper, was a little less than 19 per cent. Dr. Harrison records gains, under similar circumstances, of 29.54 per cent., 30.20 per cent. and 44.68 per cent., and without any negative results. He also makes an interesting observation on the difference in transverse strength of bars broken with "cope" or "drag" side up, both rumbled and not rumbled, showing that the release of strain by rumbling *eliminates all difference in this respect.*

I investigated the subject of tensile tests before publishing my paper, and later, in reply to a query of Mr. Wilfred Lewis, regarding tensile tests (see "Digest of Physical Tests," vol. i., p. 237, July, 1896), I said:

"I believe it is generally admitted by engineers that, in order to obtain reliable tensile tests of cast-iron bars, it is absolutely necessary to turn the pieces in a lathe (or otherwise machine them), because rough bars are always somewhat crooked and irregular in size and shape; therefore, no matter what kind of 'adjustable grips' are used, a rough cast-iron bar, when pulled, will be subject to bending or torsional stresses, rendering the records unreliable, and turning will, of course, release cooling strains irrespective of tumbling the pieces."

Since my paper was published I have received much correspondence on the subject, and records of corroborative tests; also some interesting developments which I have been asked to regard as confidential. I have made many further tests, with modifications. For instance, when companion-sets of test-bars are cast in different moulds from one ladle of iron, one poured *hot*, the other *dull*, the hot-poured bars are, in my experience (though the contrary has been asserted in print), invariably stronger; but if the dull-poured bars are tumbled in a barrel, they gain in strength more than sufficient to overcome the loss by dull pouring.

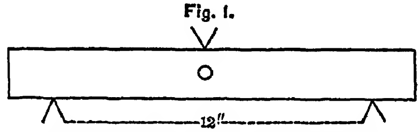
I refer to the transverse breaking of rough bars, but Dr. Harrison, after receiving this statement from me, got the same result with tensile-tests, as may be seen in the *Iron Trade Journal*, January 28, 1897, page 9.

Drilling Holes in Test-Bars.—I have always noticed that a very minute flaw or blow-hole in a test-bar decreases the transverse strength much more than can be accounted for by the area of the flaw; and the query arose in my mind, What effect upon transverse strength will drilling a hole of a certain size, across the section, at the point of rupture, have?

The *a priori* inference would be that strength would be decreased at least in proportion to amount of metal removed, or to the area of the hole.

An opposite result has been observed in many cases, viz.: If a hole $\frac{3}{16}$-inch in diameter is drilled across the section of a 1-inch square bar at point of rupture, as indicated in Fig. 1, the bar will show (in the majority of a large number of experiments)

Fig. 1.



Transverse Test Bar Drilled Through Point of Rupture.

higher strength and greater deflection than a companion-bar cast in the same mould, not drilled. In several actual instances the bar broke through solid metal at one side of the hole. In some tests two holes were bored, $\frac{3}{16}$-inch in diameter, and $\frac{1}{2}$-inch apart, and bars broke through solid metal at one side of the two holes. The gain in strength shown in these tests is not so remarkable, or so uniform, as is the gain in transverse strength of test-bars which have been tumbled in a barrel, or otherwise subjected to shock, compared with companion-bars not so treated; but I attribute the gain to the same cause, viz.: release of internal strains.

When bars are tapped on end, instead of being put in a tumbling-barrel, care must be exercised to see that the shocks *are absorbed by the bars* and not by clamps or vise, or other holders.

The following table gives the results of some tests made on

different dates with bars, 1 inch section, 15 inches long, broken on a transverse testing-machine with supports 12 inches apart; one bar being drilled across the section, the companion-bar, from the same mould, not drilled:

	Break and Strain. Pounds.	Deflection. Inches.
1. Close-grained iron, $\frac{3}{16}$-inch hole through section in middle, ...	2775	.16
2. Close-grained iron, *not drilled*, . . .	2625	.15
3 Open-grained iron, $\frac{1}{4}$-inch hole through section in middle, ...	2275	.14
4. Open-grained iron, *not drilled*, . . .	2275	.14
5. Open-grained iron, $\frac{3}{16}$-inch hole through section in middle, ...	2150	.15
6. Open-grained iron, *not drilled*, . . .	2025	.13
*7. Close-grained iron, $\frac{3}{16}$-inch hole drilled through section in middle, . . .	2550	.15
8. Close-grained iron, *not drilled*, . . .	2600	.15
9. Close-grained iron, $\frac{3}{16}$-inch hole drilled through in middle, ...	2500	.14
10. Close-grained iron, *not drilled*, . . .	2450	.14
11. Close-grained iron, $\frac{3}{16}$-inch hole drilled $\frac{3}{16}$-inch from top of bar, . . .	3125	.18
12. Close-grained iron, *not drilled*, . . .	2950	.15
13. Open iron $\frac{3}{16}$-inch hole drilled $\frac{3}{16}$-inch from top of bar, ...	1975	.13
*14. Open-grained, *not drilled*, ...	2050	.13
15. Open grained, two holes, $\frac{3}{16}$-inch in diameter, $\frac{1}{2}$-inch apart, through middle of the bar, fracture occurred between the holes, ...	2075	.13
16. Open-grained, *not drilled*, ...	2075	.13

The Effect of Vibration upon the Structure of Wrought-Iron.

Continued Discussion (see Vol. xxiii., 143, 557, 560, 573, 574, and xxiv., 809).

(Pittsburgh Meeting, February, 1896.)

EDWARD B. GRUBB, Burlington, N. J.: In connection with the discussion relative to the effect of vibration upon the molecular structure of wrought-iron, I would like to call your attention to the following extracts from a work by William

* Nos. 7 and 14 broke through solid metal.

Fairbairn. They will be found in *Iron, its History, Properties, and Processes of Manufacture*, Edinburgh, William and Charles Black, 1869.

After speaking of the effects of heat and magnetism upon the structure of wrought-iron, the author refers to the breakage of an axle on the Versailles railway, and says in this connection, on page 238:

"Undoubtedly the broken axle presented a crystalline fracture; but it has never been ascertained how far heat and magnetism were in operation, as in the case of an axle, and more especially a crank-axle, the constant vibration caused by irregularities in the way and the weight of the engine, appears to be quite sufficient to occasion the breakage without aid from the other forces. Undoubtedly, in almost all cases of the sudden fracture of axles or wrought-iron bars, during employment, the fracture presents a crystalline structure; but we believe that any molecular disturbance, such as impact, can effect this by breaking the fiber into a number of prisms, each of which, carefully examined, has the appearance of a crystal; the only question being, How long will the material sustain the repeated effects of strain-action before it breaks?"

On page 239, Mr. Fairbairn says:

"Arago and Wollaston have paid considerable attention to this subject, the latter having been the first to point out that native iron is disposed to break in octahedra and tetraheda, or combinations of these forms. The law which leads to fracture in wrought-iron from changes in the molecular structure, operates with more or less intensity in other bodies; repeated disturbances in turn destroying the cohesive force of the material by which they are held together. A French writer of eminence, Arago, appears to consider the crystallization of wrought-iron to be due to the joint action of time and vibration; but we think, with Mr. Hood, that time and its duration depend entirely upon the intensity of the disturbing forces, and, moreover, that the time of fracture is retarded or accelerated in a given ratio to the intensity with which those forces are applied."

Here follows a series of tests with a wrought-iron beam or girder, subjected to vibratory strains. In conclusion, Mr. Fairbairn says, on page 244:

"It is, however, evident from these experiments, that time is an element which enters into the resisting powers of materials of every description, when subjected to a continued series of changes. These may be very minute; but assuming them to be of sufficient force to produce molecular disturbance, it then follows that rupture must eventually ensue."

It would seem, from these extracts, that Mr. Fairbairn was disposed to admit the crystallization of wrought-iron if subjected to vibration.

It may be of interest to say that I have spoken upon this sub-

ject to different men associated with the practical use of wrought-iron. They all say that vibration will ultimately change the molecular structure of wrought-iron.

R. W. RAYMOND, New York City: Mr. Grubb's citations are advanced to prove that Fairbairn "was disposed to admit the crystallization of wrought-iron if subjected to vibration." If they did conclusively show that, it would be scarcely necessary to offer them in this discussion, for the view of Fairbairn has been already stated (*Trans.*, xxiv., 811) in a quotation given by Mr. Argall, on the authority of Mr. Howe, who speaks in his *Metallurgy of Steel*, page 198, of Fairbairn's "famous assertion, 'We know that, in some cases, wrought-iron, subjected to continuous vibration, assumes a crystalline structure.'" This is a much more explicit statement than those which Mr. Grubb cites, and the difference is so significant as to make his quotations a pertinent and valuable addition to the inquiry. For it is, in view of Mr. Howe's peculiar fullness and accuracy in footnote-references to authorities, a noteworthy circumstance that for the "famous assertion" of Fairbairn, which he places in quotation-marks, he gives no reference whatever. The inference is very strong that he was not able to find the original source, and that the "famous assertion" was taken second-hand, on the authority of "fame." It would be an interesting illustration of the untrustworthy character of such indirect testimony, if it should turn out that the passages quoted by Mr. Grubb are the basis of the general impression as to Fairbairn's opinion, now current as his "famous assertion." If this be not the case, it would be important to know the date of the "assertion;" for the quotations from the book of 1869 *contradict it.*

In the first of these passages, Fairbairn declares his belief that the presentation of a crystalline fracture can be effected by any molecular disturbance, such as impact, "by breaking the fiber into a number of prisms, each of which, carefully examined, has the appearance of a crystal." In other words, he believes that whatever crystalline structure is exhibited by the fracture is simply the result of breaking up, not of rearranging, the structural elements already present.

In the second passage, Fairbairn says Wollaston was the first to point out that native iron is disposed to break into octahedra, etc., and describes the law "which leads to fracture in

wrought-iron from changes in the molecular structure," as operating by "repeated disturbances in turn destroying the cohesive force of the material by which they are held together." Moreover, he says this law is exhibited in other bodies also. Taken as a whole, the passage implies beyond escape the conception that iron and "other bodies" are made up of structural units, held together by some material, and that the effect of repeated "disturbances" weakens their cohesion and thus reveals them as individual elements. In other words, the original structure is not reconstructed, but pulled apart.

In the third quotation, the only words which bear upon this question are those which describe the "continued series of changes" producing final rupture as "of sufficient force to produce molecular disturbance." Here, as elsewhere in the literature of the subject, an ambiguity results from the loose sense in which the term "molecular" is employed. The other passages clearly indicate that Mr. Fairbairn means by "molecular disturbance" any disturbance of the interior elements of a mass not producing visible effects on the mass as a whole. In other words, if a mass consist of crystals, held together by other material (or even by simple interlocking and adhesion), anything which weakened their union and began to separate them would be, in his general phraseology, a "molecular" disturbance. That is to say, progressive internal fracture would be "molecular." He is evidently not thinking of the molecules *constituting the crystals or structural units* as rearranged by such disturbances as he contemplates.

It is in the light of these necessary inferences that we must, in my judgment, interpret the phrase, "crystallization of wrought-iron," used by Fairbairn in the second passage above quoted, in stating the view of Arago. I do not doubt that he accepts the phrase without objection, but I think he so accepts it as a convenient way of characterizing briefly the phenomenon of a crystalline appearance of fracture.

However, it is not important to prove whether Fairbairn did or did not, in 1869, or earlier, believe in the production by vibration of a rearrangement of molecules, forming crystals where crystals had not previously existed. These quotations seem to me to relieve him from that error. But, if he held it, he was in good company. It is really since 1870 that the sub-

ject has been thoroughly and experimentally examined; and while the observations of earlier date are worthy of critical examination, the views of the observers can carry no higher authority than their facts import.

The same is still more emphatically true of the impressions of practical operators. Mr. Grubb's report of the testimony of many persons familiar with the use of wrought-iron proves only that an exploded theory is still very widely held, to explain an unquestioned fact.

"Practical" men are perhaps not likely to be affected by theoretical arguments. But the arguments on this point are such as they ought to understand. The "crystallization"-theory, "practically" stated, is that individual elements, having faces like crystals, are gradually produced in a mass of wrought-iron by continued vibration with shock; that they are revealed by the fracture; and that they did not exist in the mass before the series of shocks and vibrations which have produced them began.

Now, actual experiment has failed to produce such a change. But, leaving that question apart, there is a "practical" test which ought to be conclusive. Namely, the existence of a "crystalline" structure can be revealed in a second by the sudden fracture of the iron. Is it supposed by practical men that, in this case, the formation of "crystals" has taken place in a second?

I am quite aware that the structure exhibited by a quick fracture is declared by some to be unlike that exhibited by a fracture due to long-continued shock-vibrations. I will not argue that question here. What I wish to point out is that the phenomenon involves, in the one case as much as in the other, the apparent loss of fibrous structure. In the latter, the fibers break, as Fairbairn says, "into prisms, each of which, carefully examined, has the appearance of a crystal." The language is equally true of the former. "The fibers break"—that is the whole story in either case; or else "crystallization" must be invoked for both.

Determinations of Phosphorus in Steel.

Continued Discussion of the Paper of Mr. Thackray (see Vol. xxv., pages 370 and 1012.)

(Colorado Meeting, September, 1896.)

Edward K. Landis, Philadelphia, Pa.: In studying Mr. Thackray's paper it seemed that a critical comparison of the results from different methods therein reported would be of interest. For this purpose Table I. was constructed by grouping all determinations made by the same method, and the probable errors were calculated. In this table all chemists titrating with permanganate (A^1, E, F, G and O of Mr. Thackray's paper) are classed under "Emmerton;" those titrating with nitric acid (C, J, S and Y) under "Handy;" those re-

Table I.

	Method.									
	Emmerton.		Handy.		Acetate.		Gravimetric.		Yellow Precipitate.	
Phosphorus Determined in Thousandths Per Cent.	45	80	50	80	49	80	51	85	52	85
	53	86	53	86	49	81	52	88	52	86
	51	85	46	82	46	78	49	84	49	85
	50	83	48	78	......	......	55	85	49	83
	50	84	......	......	......	......	46	80	49	83
	......	......	......	......	......	......	47	81	46	76
	......	......	......	......	......	......	50	90	47	80
	......	......	......	......	......	......	51	83	48	79
Average.......	49.8 *a.*	83.6 *b.*	49.25 *c.*	81.5 *d.*	48 *e.*	79.7 *f.*	50.125 *g.*	84.5 *h.*	49 *i.*	82.125 *k.*

Probable error of result: *a*, ± 0.0008896; *b*, ± 0.000694; *c*, ± 0.00107; *d*, ± 0.00115; *e*, ± 0.000674; *f*, ± 0.000473; *g*, ± 0.000675; *h*, ± 0.0007959; *i*, 0.0005098; *k*, ± 0.0008303.

Probable error of single determination: *a*, ± 0.001989; *b*, ± 0.001547; *c*, ± 0.00214; *d*, ± 0.0023; *e*, ± 0.00116; *f*, ± 0.000818; *g*, ± 0.0019; *h*, ± 0.00224; *i*, ± 0.00143; *k*, ± 0.00234.

The formula for calculating the probable error is $0.6745\sqrt{\frac{S}{N(N-1)}}$, where S is the sum of the squares of the differences from the mean and N is the number of determinations.

dissolving and weighing as pyrophosphate (B, D, H, J, K, R, V and Z) under "Gravimetric;" and those weighing directly (B[1], L, M, P, Q, U, W and X) under "Yellow Precipitate;" A, N and I used the acetate method.

On comparing the probable errors of the results, it will be noticed that Emmerton's and the acetate method give smaller probable errors as the amount of phosphorus increases, while the other methods do not. Other things being equal, this alone would indicate superior accuracy for these two methods.

The amount of MoO_3 per gramme of sample varies from 5.355 grammes (Dudley) to 0.4848 gramme (Emmerton). This difference does not appear to affect the accuracy of the results, and seems to be entirely unnecessary. Dr. Dudley adds enough MoO_3 to convert all the iron present into iron molybdate and still have enough left to bring down the phosphorus. The formula for iron molybdate, as given by Dr. Dudley, is Fe_2O_3, 4 MoO_3, 7 H_2O, and, therefore, each gramme of Fe would require 5.143 grammes of MoO_3 to form this compound. According to Blair and Whitfield, 1 part of phosphorus corresponds to 55.8 parts of MoO_3 in the yellow precipitate, and 1 gramme of the sample containing about 0.083 per cent. of phosphorus would require 0.0463 gramme of MoO_3 to combine with the phosphorus. From these figures it will be seen that (allowing 99 per cent. of Fe), using 0.4848 gramme of MoO_3 per gramme of sample, only 8.6 per cent. Fe is in the form of iron molybdate.

TABLE II.

MoO_3 per gramme of Sample. Grammes.	Fe as Iron Molybdate. Per cent.
3.57	70.
2.00	40.
1.5	30.
0.4848	8.6

In each case the amount of MoO_3 required to form the yellow precipitate has been subtracted, and the remainder calculated into iron molybdate. The accuracy of the results does not appear to be affected in any manner, and, therefore, it seems probable that the formation of iron molybdate is not essential. The above results would surely indicate that 0.5 gramme of MoO_3 per gramme of sample is ample when work-

ing on low-phosphorus irons and steels, and that 5.35 grammes should be enough for 1 per cent. of phosphorus when 1 gramme is taken for analysis. The amount of free acid varies; as some chemists neutralize, some boil off the excess of acid and some do neither.

TABLE III.

Neutralize.		Boil Down.		Do Neither.	
Chemist.	MoO_3.	Chemist.	MoO_3.	Chemist.	MoO_3.
A..............	0.4848	J..............	2.00	F..............	1.50
B..............	1.0710	K..............	1.00	G..............	5.35
C..............	1.4250	M..............	1.35	H..............	3.57
D..............	0.6000	P..............	1.71	I..............	2.00
E..............	0.4848	U..............	1.93	Q..............	1.71
O..............	1.8000	W..............	1.50	S..............	2.25
Z..............	1.5750	X..............	1.50		
Average......	1.053	Average......	1.57	Average......	2.73

The MoO_3 given in Table III. is reduced to the basis of 1 gramme of sample.

It will be noticed that the greater the amount of free acid the greater the quantity of MoO_3 taken. This is probably a coincidence, although it may be possible that the presence of free acid hinders the precipitation unless a large excess of MoO_3 is present. By neutralizing it is possible to effect a large saving in MoO_3, which is quite an advantage, as this material is rather expensive.

The conclusions drawn from the foregoing are:

1. That the acetate and Emmerton's method are more accurate than the others.

2. That Dr. Dudley's phosphorus-factor is wrong and Emmerton's is correct.

3. That it is not necessary to have enough MoO_3 present to form iron molybdate with all the iron and leave an excess, after deducting enough to combine with the phosphorus.

4. That it is advisable to nearly neutralize before precipitation.

I should like to ask Dr. Dudley to state his reasons for using ten times the amount of MoO_3 which is necessary, and what authority he has for iron molybdate and for using the factor

1.9 instead of 1.794. Blair and Whitfield have shown* that the ratio of MoO_3 to phosphorus in the yellow precipitate is 100 : 1.794, and I would like Dr. Dudley to show where they are wrong.

If the sample used by Mr. Bachman† is still in existence it would now be in order to re-determine the phosphorus in it by Emmerton's method and compare the result with the figures in Mr. Bachman's paper, read in 1882, before the invention of volumetric methods for phosphorus.

The Cycle of the Plunger-Jig.

Communications in discussion of the paper of Prof. Robert H. Richards (see p. 3).

(Pittsburgh Meeting, February, 1896.)

HENRY LOUIS, Newcastle-upon-Tyne, England (communication to the Secretary): I think very highly of the novel and ingenious device of Prof. Richards for analyzing the movement of the various elements of the jig; it is a method that certainly seems to promise valuable results, if we could only make certain that these various indicator-appliances have no effect upon the action of the jig. I would like, however, to suggest that in future experiments the connection between the plunger indicator-rod and the pencil should be made by means of parallel motions, so as better to reproduce the rectilinear motion of the piston. I should expect, if this were done, that the "curve" of the plunger in the Harz jig, at any rate, would be represented either by straight lines or else by quite regular curves, according to the method by which the plunger is set in motion; the apparent combination of straight line and curve seems difficult to explain.

With respect to the water-curve, I am afraid I am completely at fault, but no doubt Prof. Richards can explain my difficulty. In Fig. 2, the areas of the screen-side and the plunger-side of the jig are equal. If the piston be a fair fit and the bottom of

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* *Jour. Am. Chem. Soc.*, vol. xvii., p. 759, October, 1895. See also Mr Blair's remarks in discussion of Mr. Thackray's paper, *Trans.*, xxv., 1012.

† *Trans.*, x., 322.

the jig (Fig. 1) be closed, I cannot see why the velocity of the water should be less than that of the plunger, or where the water goes to that is not forced through the sieve; *e.g.*, in Fig. 2 the plunger should have displaced ($4.75 \times 24 \times 48 =$) 5472 cubic inches.* On the other hand, the amount of water actually displaced, as shown by the water-curve, is only ($2.15 \times 24 \times 48 =$) 2476.8 cubic inches, or less than half. I should like to have Prof. Richards state where the rest of the water has gone. It has not gone over the overflow; otherwise the curve would have a long, flat top.

The quartz-curve starts somewhat as I should have imagined it would, showing a gradual acceleration, which becomes uniform motion in apparently 0.033 seconds—a very short time, as it seems to me. This certainly does not bear out my *a priori* conclusion that the particles in a jig would not, as a rule, have time to reach uniform velocity, but fairly disproves that view. The motion then continues uniform; but it is another surprise to me to find that both the quartz-curve and the ore-curve have attained their maxima and have commenced to descend, before the water has done so. The water-curve reaches its highest point, as I should expect it to, just a little after the plunger itself has done so; but why the heavy particles should commence to fall while the water is still moving up with practically its full velocity, instead of just after the water has commenced to fall, is to me so inexplicable that I prefer for the present to believe in some imperfection in the co-ordination of the results or in the working of the instrument, rather than accept the diagram here given. The rate of rising of the quartz, when it has become uniform, is slower than that of the water, as it should be; but it is surely rather surprising to find their rate of falling practically equal, until the quartz again reaches the bed. Again, the ore rises more slowly than the quartz, but its rate of falling is likewise practically uniform with that of the quartz and of the water; in fact the three lines W, R and O are sensibly parallel in the downward portion of the curve. The distances through which the three move are respectively 2.05 : 1.16 : 1 and the times occupied as 2.11 : 1.11 : 1, or in other words the falling velocities are just about equal. According to

* I have scaled the stroke of the plunger on the diagram, which shows $4\frac{3}{4}$ inches.

the data before us here, the quartz and ore, having fallen to the sieve-level, remain there until the next pulsation again lifts them; this present investigation does not show the effect of suction at all as far as the minerals are concerned; it merely shows that the downward current of the water continues with uniform velocity whether the bed of minerals be moving or at rest. Ought not the effect of suction to be rendered visible by the O line falling somewhat below the datum-line in the last part of the stroke?

I have confined my remarks to the first card given by Prof. Richards, because I have found so many difficulties in it that I am afraid there is little use in going on until these are partially, at any rate, explained. Would it not have been as well to have made a few experiments upon a jig full of spheres of two different substances, thus having all the particles of equal diameters and of different specific gravities, and to have noted their different behaviors at varying piston-speeds? I know that this would be further removed than Prof. Richards's experiments from the conditions of actual practice; but, like Prof. Richards, I am now concerned only with the question as a pure theoretical speculation.

I am sorry that my criticisms should be so continually taking a destructive tendency; it is certainly not because I am insensible to the real merits of Prof. Richards's work. He has, it seems to me, made a great step forward with his new instrument, which will yield important information when we understand it better. I have been trying to devise an apparatus for getting similar results photographically, but so far without success. I found so many difficulties in the way that I can the better appreciate the ingenuity with which Prof. Richards has taken this first step towards dissecting the actual action of the jig. I still think that a photographic method would be the best, though I must confess that I do not yet see how to arrange it.

Prof. Richards (communication to Secretary): In replying to the criticisms of Prof. Louis, I will take up the points one by one in the order that they occur in his remarks.

1. His suggestion of parallel motions for the pencil-rods does not meet my approval: first, because it would greatly complicate an instrument which must be simple in order to succeed; and secondly, because it is unnecessary, since the pencil-rods

run between anti-friction rollers and I have many times watched the cork-float rising and falling with the water, when connected with the instrument and when disconnected, and can see no difference in its behavior.

2. With regard to the kind of curves that are produced, I would say that, if the Harz jig obeyed a simple mathematical law, uncomplicated by the many disturbing influences which occur (for example, leaky plunger, vibrations of the machine, secondary vibrations of the water, inequalities in the bed, bending of the excentric shaft, back-lash in the excentric), then all my plunger- and water-curves would be the projections of screw-threads or helixes. The plunger-curves approach this form in the majority of cases. Fig. 6 may be possibly a trifle better than its neighbors in this respect. The central portion of a long-drawn-out helix will appear to be a straight line; the ends will appear to be curved.

3. The difference between 5472 cubic inches displaced by the plunger, and 2477 cubic inches of water raised, which seems large, is, I think, attributable to three causes:

a. The water-curve is much flattened on the lower end. The cork was resting on the rock while the water disappeared from view, coming up again on the return of the plunger.

b. The plunger in this case, as in all Harz jigs in this country, does not fit its box, but has quite a leak all round, particularly on the sides, to allow for the rocking-movement of the excentric-wheel. This leak will shorten the water-curve as compared with that of the plunger considerably.

c. The screen is of $\frac{3}{8}$-inch round holes and $\frac{3}{8}$-inch space. The rows of holes are at right angles to each other both ways. If we draw a square with its four angles at the centers of four contiguous holes, we see that for 0.11 square inch of hole there will be 0.45 square inch of solid metal. The holes occupy only 19.6 per cent. of the total area. This excessive contraction increases largely the friction, the leak of the plunger, and hence the diminution of the curve.

4. I think the return of the quartz and ore slightly before that of the plunger and water is as it should be; for the upward speed of the water slackens as it approaches its apex, and allows the quartz and ore to fall. Some of the curves fail to show this quality; for example, Figs. 7 and 10, where the rock is even a

little later than the plunger. This is probably due to an error in combining the curves. The quartz and ore will always start downwards slightly before the water does. The quartz in its descent has reached a higher velocity than the water before the former came to rest on the bed. The distance of its fall is so short that this point is not strongly marked.

5. The lines W, R and O are not parallel. If we measure the distance between W and R on the ordinate which is $2\frac{1}{4}$ inches from the left edge, we find it to be 1.25 inches; if we measure the distance between W and R on the ordinate which is $2\frac{1}{2}$ inches from the left edge, we find it is 2.45 inches, showing that the quartz is falling faster than the water.

6. It is at present only possible to exhibit the conditions when and where the suction takes place. It is impossible to attach a thread to a fine particle and have it record its doings while its coarser neighbors are doing their duty as bed. This *when* and *where* is very clearly shown by most of the jig-curves given in the paper.

7. It is unfortunate that Prof. Louis chose the most extraordinary jig-record in the whole paper for his discussion. I would suggest to him that he examine Figs. 4, 5 and 11 for plain excentric jigs, and 25, 26 and 27 for accelerated jigs. I think these are as free as any from instrumental errors or abnormal jig-actions.

8. With regard to testing the jigging-operation upon spheres of equal diameter and of varied specific gravity I will be very glad to lend my instrument to make such a test, if Prof. Louis or any other competent engineer desires to pursue the investigation in that direction.

I am quite sensible of the fact that the instrument has its defects, the greatest of which is, that the quartz- and water-cards are not taken at the same time. This defect renders it necessary to combine two cards taken at different times; and the personal equation comes in to some extent in that work.

I hope in the near future to rearrange the instrument so that, for a jig of moderate plunger-throw, the two cards can be taken together.

The Accumulation of Amalgam on Copper Plates.

Discussion of the Paper of Mr. R. T. Bayliss (see p. 33).

(Pittsburgh Meeting, February, 1896.)

L. D. GODSHALL, Everett, Washington: This very interesting paper cannot fail to command the attention of every one who has ever had experience in the amalgamation of gold-ores. I wish to call attention to three very important facts brought out very prominently by the paper, namely:

1. The extent of accumulation of amalgam on copper plates in the form of scale, not removable in the ordinary manner;
2. The change in the relative quantity of the two metals in the amalgam as we recede from the battery-discharge;
3. The difference in fineness of the bullion obtained from the above-mentioned scale as compared with the bullion from the daily clean-ups.

From Mr. Bayliss's figures we find that during the life of the plate in question 14,942 tons of ore were crushed, yielding amalgam of the value of $5.03 per ton, or a total value of $75,-158.26. The amalgam accumulating on the plate during this time as scale, we find to have had a value of $8340.54, or $0.558 per ton of ore treated, representing 11.09 per cent. of the total amount of amalgam saved in the ordinary clean-up; or, taking the amalgam saved by daily clean-up plus the amalgam-scale as representing the total saved on this plate, or 100 per cent., we find that the daily clean-up saved 90.01 per cent., while the scale represented 9.99 per cent. The above figures do not include some additional amalgam still adhering to the plate after scale had been removed.

We are told that this was not an exceptional case at the Drum Lummon, but that other plates have even yielded a third more than the one in question. The interesting query now comes up, whether this experience at the Drum Lummon is exceptional or universal at all mills; and another question arises, as to how long such an accumulation might continue,

and at what rate. It is to be hoped that the experience of others may be given, in order to throw additional light on this subject.

The difference in composition of the scale and ordinary amalgam is very interesting. Tabulating the results given by Mr. Bayliss, we obtain the following:

Average Fineness of Bullion Obtained from:

	Amalgam of Daily Clean-up.	Scale.	Amalgam from Copper Plates on Frue Vanners.
Au	541.5	431.4	380.5
Ag	413.9	562.5	602.0
Total	985.4	993.9	982.5

The above shows conclusively what Mr. Bayliss claims, "that the gold-fineness of bullion is highest nearest to the battery, and gives place to a steadily increasing silver-fineness as the amalgam is deposited upon the copper plates at greater distance from the battery-discharge."

With reference to fact No. 2, there is no doubt in my mind that the change in the composition of the amalgam as we recede from the battery-discharge is caused by a difference in the tendency of the gold and silver to amalgamate, the former amalgamating more easily than the latter, and therefore more quickly, and hence being found nearer to the battery-discharge.

Fact No. 3, however, is not so easily explained. In the first place we must assume some cause for the formation of the scale. Let us call this the affinity of the amalgamated gold and silver for the amalgamated surface. I use the term "amalgamated" gold and silver, as it is evident that the gold and silver must first pass through the form of amalgam before becoming scale; the next assumption then naturally follows, that the cause of the phenomenon is a greater affinity of silver for the amalgamated (or scale-forming) surface than that of gold.

Mr. Bayliss, referring to native silver in the ore, claims that the facts in the case disprove the theory which ascribes the results to such native silver. He does not, however, take into consideration that the condition of the native silver in the ore

may be very different, or its behavior very different, from that of the same silver after amalgamation. In that distinction, to my mind, lies the solution to the problem.

Arthur L. Collins, Central City, Colo.: The interesting paper of Mr. Bayliss recalls a recent experience of my own, which may be useful as a comparison.

We had occasion recently to rebuild the Hidden Treasure mill, of 75 stamps, at Black Hawk, Colo., after nearly 10 years of service. This mill also belongs to an English company—the California M. & M. Co., Ltd.,—operating the California mine, Gilpin county, Colo. It is a typical Colorado mill, with roomy mortars, high drop and deep discharge; two inside plates, and an apron-plate 54 inches wide by 12 feet long, being used for each battery of 5 stamps. The ores treated are soft, consisting of quartz and decomposed feldspar, carrying from 10 to 20 per cent. of sulphides, mainly iron pyrites and zinc-blende. The stamping is very fine; a slot-screen equivalent to a 50- or 60-mesh being used. The ore milled yields on an average 6 dwt. of crude bullion per ton, 740 fine in gold and 220 in silver, the balance being mainly copper. But as large amounts of custom-ore are treated at the mill, this fineness varies within wide limits.

On rebuilding the mill, each plate was thoroughly scoured for several hours with sharp sand from the tailings, hot water and quicksilver. The loosened amalgam could then be removed (after washing off the sand with hot water) with a rubber scraper in the ordinary way, leaving the plate with a good, soft surface of amalgam, in excellent condition for further use. In this way an average of $100 was saved from each plate; the crude bullion carrying the ordinary proportion of gold and silver, with a slight increase in the copper, from the scouring of the plate.

To ascertain the value still left on the coppers, 5 plates were shipped to the Argo smelting works, realizing an average of $150 each, or about 8 ounces of fine gold per plate. Thus after 10 years of constant use, and after about as much ore had passed over it as in the Montana case, reported by Mr. Bayliss, each plate yielded only $250 instead of over $8000.

It is only a rich corporation like the Montana Mining Company, Limited, which can afford to keep $8000 locked up idly on each plate, or $80,000 in a mill of 50 stamps. Without

considering interest on so large a sum, possible losses from theft or fire, or even from the abrasion or peeling off of parts of the amalgam-scale, should alone suggest some need for improvement. Such practice needs the strongest justification of absolute necessity or direct advantage—neither of which I venture to think can be shown for it.

In this district, where over half of the 500 or 600 stamps of the camp are run on small lots of custom-ore, on each of which a complete clean-up has to be made, it would be impossible to run in that way. The whole difference probably lies in the mode of cleaning the plates, and especially in the use of the steel scraper. This much-abused tool can doubtless be used so as to ruin any plate; but in proper hands it is indispensable to the thrifty millman. The scrapers used in this district are made from $\frac{1}{12}$-inch steel, about 3 inches wide, ground square across, so as to give two right-angled edges.

We find that with certain ores of this district—notably those from the Bates-Hunter, Fisk and Corydon mines—the amalgam has a strong tendency to form hard scale on the plates, which cannot be removed with a rubber scraper, whisk-broom or any such tool. But with a steel scraper the scale can be broken through and cracked off like a layer of dry putty, and this, if done properly, leaves the plate with a soft and "satiny" amalgam surface, if anything better for catching gold than the uneven surface of harder scaled amalgam. The use of the steel scraper is almost always necessary with inside plates, where the scouring of the heavy mineralized pulp seems to harden all the amalgam.

Experience here shows that amalgam which forms hard scale on the plates yields bullion finer in gold and lower in silver than the softer variety; and it is generally obtained from ores carrying comparatively coarse or flaky gold. Possibly only the coarser precious metal tends to form hard amalgam scale on copper plates, so that while, in the Drum Lummon ore, the native silver is the coarser, in the Gilpin county case, where native silver is almost unknown, scale can only form from ores carrying coarse gold.

One part of Mr. Bayliss's experience, as to the increasing proportion of silver to gold in the amalgam caught farthest from the battery, is duplicated here. After the apron-plate, an

amalgam-trap, and about 5 feet of blanket-sluice, we secure part of the concentrates on percussion-tables of the continuous (Gilpin county) type, the bed of the table being made of copper and amalgamated. On these a further trifling amount of amalgam is secured, and still more amalgam gradually accumulates on the tables, forming a rough scale, which is tough enough to trouble even a steel scraper, and can only be fully recovered when the table is worn out. Amalgam secured on this table yields bullion which almost always assays higher in silver and lower in gold than the average secured on the apron-plate above.

ROBERT GILMAN BROWN and R. C. TURNER, Bodie, Cal.: The facts of particular interest in Mr. Bayliss's paper are the value of the absorbed amalgam and the changed ratio of silver to gold in the adhering amalgam, as compared with that of the routine clean-ups.

A single statement on the first point deserves special consideration, namely, that "upon being cut up and melted into a bar, the value of the gold- and copper-contents amounts to more than twice the value of a new plate; hence the substitution of new plates for old is an expenditure which can be viewed without concern."

This conclusion is not of necessity true; and whether it be true or not depends upon a question to which the conditions of this case give no answer.

Does the absorption of amalgam into a new plate proceed promptly, being chiefly accomplished within, say, the first thirty days of use, or is it a direct function of the time, increasing in simple proportion?

On the face of the matter the former seems the most probable, as the appetite of the plate for the amalgam should become weakened with its partial satisfaction. On the other hand, it is possible that the soaking of the amalgam into the plate may proceed so slowly as to make the monthly rate appear practically constant for several months or even years.

To obtain a little light on this point, three strips of silvered copper, approximately 1 by 3 inches in size, were placed on the battery-apron of the Standard Company's gold-mill, at Bodie, California, which is operated without battery-amalgamation. There they were left for varying periods and subjected to

Tests of Absorption of Gold and Silver by Copper Plates in the Standard Consolidated Mill, Bodie, Cal.

Number of Strip.	Weight in Grammes.	Area in Square Inches.	Milligrammes of Silver on each Strip. Calculated from Area.	Number of Days Strip was Exposed.	Number of Milligrammes of Gold and Silver on Strips after Use.		Number of Milligrammes Gain in Silver by Use. Calculated by Areas.	Number of Milligrammes of Gold and Silver Absorbed per Gramme of Copper.		Number of Milligrammes of Gold Absorbed per Square Inch of Top Surface.	Number of Milligrammes of Silver Absorbed per Square Inch of Top Surface.	Ratio of Gold to Silver.
					Gold.	Silver.		Gold.	Silver.			
1	6.56	3.00	2.652	2	22.12	5.68	3.028	3.37	0.40	7.37	1.01	100:14
2	6.67	3.02	2.670	7	24.10	5.22	2.55	3.61	0.38	7.98	0.84	100:10
3	6.94	3.10	2.740	14	29.40	6.94	4.20	4.24	0.54	9.48	1.35	100:14
Sample assayed	0.8228	.43	0.380									
Sluice plate	0.687	....	....									
										Fineness.		
										Gold.	Silver.	
Sluice-plate amalgam										108.77	84.39	100:77
Sluice-plate										3.64	0.61	100:17

the same routine of cleaning and scraping with the aprons themselves. They were finally cleaned as closely as possible, all surface-amalgam being removed, and their values were determined. The complete results appear in the foregoing table:

These leave much to be wished for in the way of more complete information, but also serve to strengthen the *a priori* probability of the alternative first stated above.

Fig. 1.

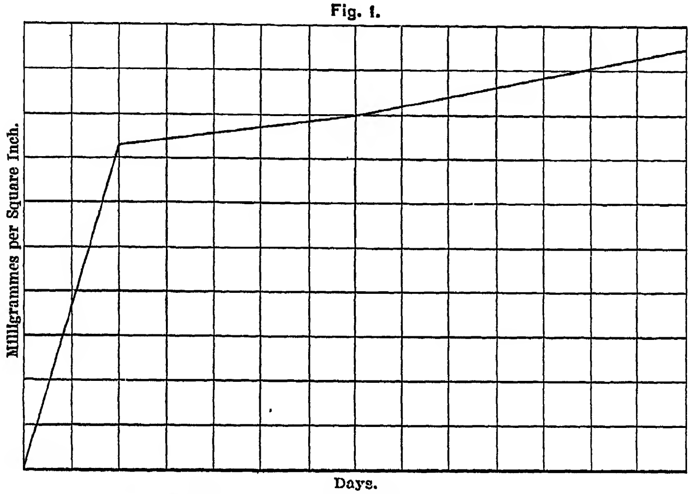


Diagram Showing Rate of Gold-Absorption.

Studying the absorption of gold only, the rate per day per square inch of surface for the several strips is as follows:

No. 1—3.68 milligrammes.

No. 2—1.14 milligrammes.

No. 3—0.68 milligrammes.

Or for No. 2, assuming its unit-increase for the first two days to be the same as that of No. 1, the increase for the remaining five days is 0.60 milligrammes, or 0.12 milligrammes per square inch per day. And for No. 3, assuming its increase for the first seven days to be the same as for No. 2, for the remaining seven days it is 1.51 milligrammes, or 0.22 milligrammes per square inch per day. This result, though not quite at one with that from No. 2, still marks strongly the tendency in the case. This appears still more forcibly in the diagrammatic expression, Fig. 1.

From this it would appear that the destruction of plates for the sake of the absorbed amalgam may be poor economy. For if, as appears probable from this experiment, the greatest absorption takes place in the first few days of the run, the larger part of the value recovered by destruction of the plate is immediately taken up again from the daily amalgam, and at the same time the mill is taxed for a new plate.

But, in any event, it would appear that the recovery of absorbed values by destruction of the plate is of the nature of a discounted loan, in which the discount amounts to the value of the new plate, and the repayment of which must be provided for by monthly deductions from product, to the amount of which the loss of interest on these deductions must be added. With more complete data as to the rate of absorption this could be expressed as a formula; but the constants would certainly vary with different ores and changed conditions of plates.

The above experiments were carried on with very soft and "wet" plates, and it is to be presumed that such conditions would favor early absorption. Certainly there is no possibility of accumulations of hard amalgam to any considerable thickness on the plates of the Standard Company's mill, and it will be admitted that a coating of $\frac{16}{100}$ to $\frac{1}{16}$-inch is, to say the least, extraordinary. It would seem that a judicious use of the steel scraper would have prevented this, with no appreciable harm to the plates; the plates in the Standard mill have been relieved of the harder amalgam with a steel scraper for four and one-half years, and are to-day in prime condition.

The second point, of the high silver-ratio of the adhering amalgam can, to a large extent at least, be traced to the stiffness of silver-amalgam, which is a very noticeable feature of it. This would resist removal by the rubber scraper more than the softer gold-amalgam.

In this connection it would have been interesting to have had recorded the ratio of silver to gold in the bar resulting from the destruction of the plate, for comparison with the ratio in the amalgam. In the experiments here recorded it is noticeable that the ratio is but 10 to 14 parts silver to 100 parts gold, while the average plate-amalgam gives about 75 silver to 100 gold. The plate then exerts a straining action, being more permeable to gold than to silver-amalgam. This is borne out by

the extra tests on an old piece of copper sluice-plate. The amalgam scraped from its surface carried gold and silver in the ratio of 100 : 77, while the plate itself showed but 100 : 17.

The whole question is one of commercial as well as scientific interest, and Mr. Bayliss should be thanked by all concerned in gold-milling for having drawn attention to the facts.

Some additional experiments have been undertaken with the view of gaining data from more extended exposures to amalgam; and these will be reported to the Institute in due time.

MR. BAYLISS: The interesting contributions to this discussion call for some notice and a few remarks in reply.

Mr. Godshall, in considering "the change in the relative quantity of the two metals in the scale-amalgam as we recede from the battery-discharge," accounts for the increasing proportion of silver on the theory that the gold in the ore amalgamates more easily, and, therefore, more quickly, than silver; but, in seeking for an explanation of the fact that the fineness of the bullion from scale-amalgam differs so widely from the fineness of the bullion from the daily clean-ups, he suggests that, after the two metals have passed into the form of amalgam, the silver therein contained has a greater affinity for the amalgamated (or scale-forming) surface than gold, and goes on to suggest that I have not taken into consideration that "the condition of the native silver in the ore may be very different, or its behavior very different, from that of the same silver after amalgamation." In this distinction he finds a solution of the problem.

I submit this distinction is a very subtle one and freely admit that I am unacquainted with the fact that the process of amalgamation reverses the relative affinity of gold and silver for an amalgamated surface, if any such difference exists.

The experience of Mr. Collins in the Hidden Treasure mill, at Black Hawk, Colo., does not appear to be quite in line with the case under review, as his mill was largely employed in the treatment of custom-ores, which, if paid for on mill-returns (as seems to be the practice), would necessitate the careful recovery of accumulated amalgam after each operation. It is, however, of interest to note that the value of Mr. Collins's plates after they had been thoroughly scoured, as determined by the Argo smelting-works, namely, 8 ounces of fine gold per plate, cor-

responds very closely with the experience in the Drum Lummon mill, where the gold-contents of a plate, after the removal of the scale, amounts to 8.96 ounces.

Mr. Collins, with some force and great fairness, challenges the practice of permitting the accumulation of amalgam as neither necessary nor advisable. I should be indisposed to defend such practice as a universal or arbitrary principle; for I am satisfied that special causes contribute to the accumulation of amalgam in the Drum Lummon mill in such a marked degree. But in this particular case the practice seems to be justified by the fact that careful records of the daily pulp- and tailing-samples taken from batteries served, on the one hand by a plate coated with this scale and on the other hand by a plate not so coated, and also, in another case, by a plate from which the scale had been removed by a steel scraper, proved beyond question that there was in the first instance an additional percentage of gold recovered, which completely outweighed the undoubted disadvantages of the practice, such as loss of interest on money, risk of possible loss from fire, etc. Moreover, the necessity for the practice in the Drum Lummon mill has been proved by the fact that it has been found impossible to remove the scale, which is extremely hard, by means of a steel scraper without injury to the plate itself and the destruction of its amalgamating-efficiency.

Messrs. Brown and Turner seem to me to have mistaken the object in view in the substitution of new plates for old after the accumulated scale has been removed, inasmuch as they point to the result of their very interesting and complete experiment on the period covered by the absorption of amalgam by a new plate, as evidence "that the destruction of plates for the sake of the absorbed amalgam may be poor economy." This impression on their part may be removed by the statement that the new plates are not substituted for the old for the purpose of recovering the amalgam still remaining on the latter, but because the drastic treatment to which the old plates have been subjected in the removal of the scale, as fully described in my paper, renders them wholly unfit for further use.

They remark also that "the high silver-ratio of the adhering amalgam can, to a large extent at least, be traced to the stiffness of silver-amalgam, which would resist removal by the

rubber scraper more than the softer gold-amalgam." In the case under discussion it is not possible to subscribe to this theory because experience tends to prove that the silver-amalgam obtained from the Drum Lummon ores is not so stiff and solid as the gold-amalgam.

The inquiry of Messrs. Brown and Turner as to "the ratio of silver to gold in the bar resulting from the destruction of the plate" reveals a strange conflict, in the case under discussion, with the results obtained by them in the experiment to test the absorption of gold and silver by copper plates. They report that the metal absorbed was in the ratio of "but 10 to 14 parts of silver to 100 parts gold, while the average plate-amalgam gives about 75 silver to 100 gold."

The copper-plate at the Drum Lummon mill, on the other hand, upon being run into a bar, was found to contain 8.96 ounces gold and 9.62 ounces silver, or in the ratio of 93 parts gold to 100 parts silver; whereas the average plate-amalgam from daily clean-ups gives 82 parts of silver to 100 parts of gold, showing, in this instance, that the plate was apparently more permeable to silver- than to gold-amalgam, instead of the reverse, as proved by the tests in the Standard Consolidated mill.

And, completing the comparison by ratio throughout each stage of the amalgam obtained from the plate in question, we find:

1. Amalgam from clean-ups contains gold 100, silver 82.
2. Amalgam from accumulated scale contains silver 100, gold 76.
3. Amalgam from vanner-plates contains silver 100, gold 63.
4. Gold and silver absorbed in plates, silver 100, gold 93.

In conclusion, I submit that we must look to chemical, and not to mechanical, causes for the explanation of the two important facts raised in this discussion, namely, the excessive accumulation of scale and the variation in the relative value of gold and silver in the amalgam taken off the plates from day to day, as compared with that remaining on them in the form of scale. It is a question which I must of necessity leave in more capable hands, but in the solution of which I shall be pleased to assist by affording all the information at my command.

Frank Owen, Marble Bar, Western Australia (communication to the Secretary): In connection with the discrepancy ob-

served by Mr. Bayliss in the relative fineness of the bullion obtained by amalgamation on copper-plates from an ore containing gold and silver, and the bullion obtained afterwards by scraping the same plates, a somewhat similar experience with a different class of ore may be of interest.

The observation to which I refer was made at the El Silencio mine, owned by the Frontino and Bolivia Gold Mining Co., Ltd., of London, and situated at Remedios, department of Antioquia, Republic of Colombia. The ore consisted of auriferous (mixed with a little arsenical and occasionally antimonial) pyrites, galena and zinc-blende. The proportion of pyrites in the ore ranged from 1 to 12 per cent. The average assay-value of the ore was 1 ounce gold and 5 ounces silver per long ton. It was mined at a depth of from 220 to 285 feet. The gangue was quartz, and the country-rock syenitic granite, often much decomposed, so that it was easily crushed. The proportion of silver to gold was found to be considerably higher in the galena than in the pyrites.

The ore was treated in a wooden 16-stamp mill (four batteries of four heads each) of the native pattern, common in Colombia and Brazil, driven by an over-shot water-wheel, and capable of crushing a maximum of 550 tons (of 2240 pounds) monthly. The shoes were of cast-iron, weighing 140 pounds, and the total falling weight was 200 pounds. The mill was run at 40 drops of 5 inches per minute. The screens were of 20-mesh diagonally-punched Russia iron. Silver-plated copper-plates, 4 feet long by 2 feet 6 inches wide, and set at an inclination of 1 in 20, were used outside the battery only, and were cleaned up once in 24 hours. The usual grade of bullion produced was gold, 600; silver, 350; total, 950. The amalgam contained from 55 to 60 per cent. of sponge-gold, which lost from 1.5 to 2 per cent. in melting.

In February, 1891, four old plates, which had been in use from 2 to $2\frac{1}{2}$ years, were scraped, and produced bullion of the value of $22,500. The amalgam obtained by scraping contained from 45 to 50 per cent. of sponge-gold, which lost from 3 to 4 per cent. in melting. The grade of the resultant bullion averaged: gold, 450; silver, 300; total, 750; whereas, judging from the proportions usually obtained from the plates, one would have expected it to show: gold, 473; silver, 277. After chip-

ping off all the amalgam as far as practicable, the plates were melted down into 30 auriferous copper bars, which realized $750.

As I do not care to "step in" where I notice that men of so much greater experience in stamp-milling "fear to tread," I shall not attempt to suggest any explanation of the problem presented by these observations.

Copper-Ores in the Permian of Texas.

Discussion of the Paper by Mr. E. J. Schmitz (see p. 97).

(Pittsburgh Meeting, February, 1896.)

HENRY LOUIS, Newcastle-upon-Tyne, England (communication to the Secretary): I have been much interested in Mr. Schmitz's description of the copper-ore bed in the Permian formation of Texas. He compares it with the Mansfeld *Kupferschiefer;* but it seems to me to present closer analogies with a small deposit on the same side of the Atlantic, in Nova Scotia, which I examined in 1877. I quote entirely from my field-notes, made then, and from specimens still in my possession. These copper-deposits are known in several places; but I examined them at the village of New Annan, on the banks of the French river. This district has been determined by Sir J. W. Dawson as being of Permian age, and this determination seems to have been confirmed as recently as 1891 by the operations of the Canadian Geological Survey. The following is the section, very imperfectly exposed in a few shallow levels, described in ascending order:

Red sandstone, moderately fissile and markedly false-bedded.

Lower nodule-bed, 1 inch to 6 inches thick, of fissile micaceous sandstone, containing obscure plant-remains converted into anthracite, chalcopyrite, chalcocite and iron pyrites.

Very coarse grit or, in places, red sandstone; thickness and character variable.

Upper nodule-bed, 6 inches to 2 feet thick exceptionally, generally about 10 inches.

Soft, gray, friable shaly sandstone, with nodules of copper-ore.

Red sandstone.

All these strata lie flat, in places quite horizontal, in places dipping towards the river at 5° to 10°.

The copper-nodule beds are exposed in two places, about a mile apart; but it is doubtful whether they are continuous over the area.

The upper nodule-bed is the more important one; the nodules vary in weight generally from ¼ ounce to ½ pound, but have been found up to 1½ pounds. They consist of very pure chalcocite, but I also found some of covelline, more or less pure, and of other indeterminate copper minerals, possibly all alteration-products of chalcocite and chalcopyrite.* All the nodules were coated with carbonate of copper, which had also filled up any cracks in the nodules, and had impregnated the sandstone bed to a certain extent.

It will be seen that the occurrence presents many striking analogies with those described by Mr. Schmitz, while the main point of difference is no less striking, namely, that his nodules are chiefly carbonates and silicates, whilst mine were sulphides. It would be interesting to know whether he was in any case able to detect kernels of sulphides in these nodules, or whether it must necessarily be supposed that they were formed originally as carbonates and silicates. Deposits of these oxidized ores seem almost invariably to be of secondary origin, so that it becomes important to know whether these Texas Permian deposits are exceptions. In the New Annan deposits, pebbles were everywhere conspicuous by their absence, as seems to be the case also in Texas. In spite of the curious and persistent occurrence of copper-ores in Permian formations all the world over, I am not inclined to ascribe to these nodule-deposits any contemporaneous origin, but believe them to have been produced much later than the formation, and perhaps even partial consolidation, of the beds. The part played by the wood-remains is doubtful; these may, of course, have acted as reducing-agents, precipitating the copper-ores from a solution that was percolating through the porous beds. If so, the copper was probably dissolved as a sulphate, and the probability of the original deposition of these ores as sulphides grows stronger. The conversion of chalcocite into covelline is, of course, a comparatively well-known phenomenon.

* Analyses by myself were published in the *Transactions of the Nova Scotia Institute of Natural Science*, vol. iv., 1878, p 424.

Vein-Walls.

Discussion of the Paper of Mr. T. A. Rickard (see p. 193).

(Colorado Meeting, September, 1896.)

R. G. Brown, Bodie, Cal.: Mr. Rickard's paper, with its valuable illustrations, brings into fresh clearness the game of hide-and-seek which the miner must play with his ore-deposit; but, more than this, it goes far towards classifying those perplexing idiosyncrasies which, while unclassified, constitute a labyrinth almost without clue. Probably every engineer can add to the list by the mention of at least one fresh detail from his own experience, and I for one would like to comment on two of Mr. Rickard's divisions.

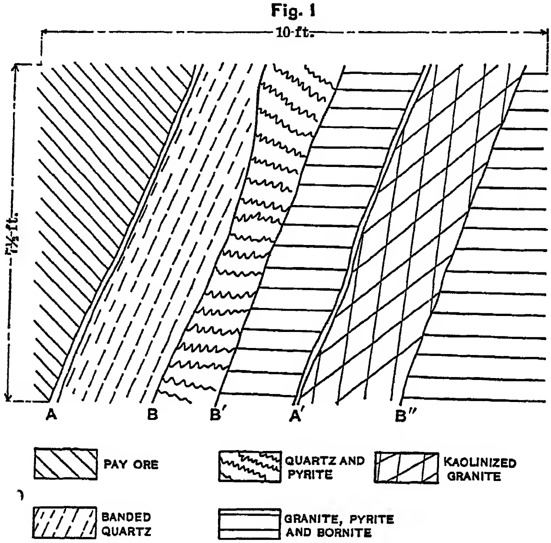
To what he has written about "walls within walls" and "walls without walls," the accompanying Figs. 1 and 2 may be appropriate, the more so as they represent conditions obtaining in a part of the Butte copper-belt more typical of the copper-veins as a whole than the Gagnon, which, in some respects the most interesting mine in the camp, stands almost alone in many particulars. (See *Trans.*, xvi., 62.)

Fig. 1 was sketched in January, 1895, in the stopes above the 700-foot level of the Colusa-Parrot mine, east of the Parrot and southwest of the Anaconda. The stope, the commercial foot-wall of which is shown at A, was 30 feet wide at this section, the pay-ore being massive chalcocite, jumbled with bornite, pyrite, quartz and thoroughly broken-down granite. A and A′ are inch-thick seams of black talc; B, B′ and B′′ are also talc, possibly ⅛-inch thick. In the plane of the section the "banded quartz" and "quartz and pyrite" were both of too low grade for extraction, but 5 feet back along the stope had been of fair value. The granite with quartz and bornite between B′ and A′ was pay-ore, but that to the right of B′′ was not. In both cases the ferro-magnesian minerals had been removed, leaving, where not obscured by ore, faint, dark patches. Between A′

and B″ the decomposition had left a gray mass of kaolin, so soft that the prong of a miner's candlestick could be thrust into it to full length.

It seems probable that not one of the walls shown was the true foot-wall of the shear-zone occupied by the ore, but that this title belonged to a considerable mass of "talc," several feet to the right, beyond which the decomposition was inconsiderable. Evidently, in veins of this kind, the most thorough cross-cutting is in order, and it is interesting to note that, in

Fig. 1



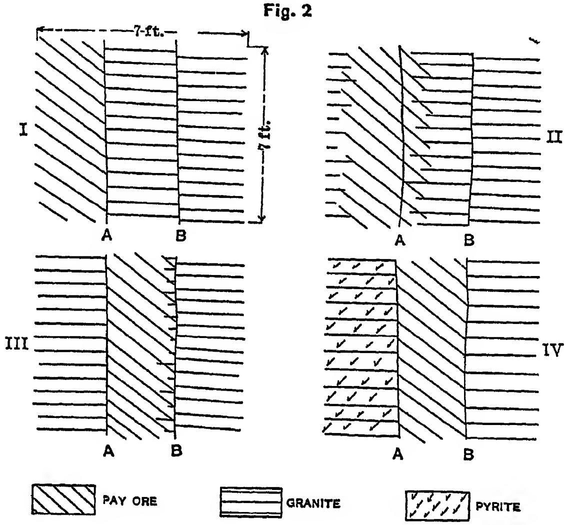
this case, A was for some time considered to be the limit of the ore.

The various portions of Fig. 2 were also drawn in January, 1895, as the work progressed, in the stopes over the 600-foot level of the same mine. I., II., III. and IV. represent faces seen in successive visits to the mine, and are approximately 5 feet apart. The figure, though introduced under the head of multiple walls, illustrates, as a whole, the gradual changing of values from one portion of the vein to another, and, finally, gives strong support to the gradual-replacement theory, which, with all its difficulties, I think fits the Butte conditions most closely.

The granite was decomposed to the extent of having its mica

represented by pearly spots, the hornblende by dark splotches and the feldspars milky. The pay-ore was chalcocite and quartz, and the pyrite occurred in fine, brilliant grains of inappreciable assay-value. Block AB showed in Section II. massive chalcocite on its left-hand edge, shading into quartz and chalcocite (perhaps 15 per cent. of the latter) a few inches in. The left-hand edge of B in Section III. was granitic in character, but fully mineralized with chalcocite. Here, again, it would have been most natural to consider A the wall of the ore, and on arriving at Section III. to have stopped, believing the ore-chute to have

Fig. 2

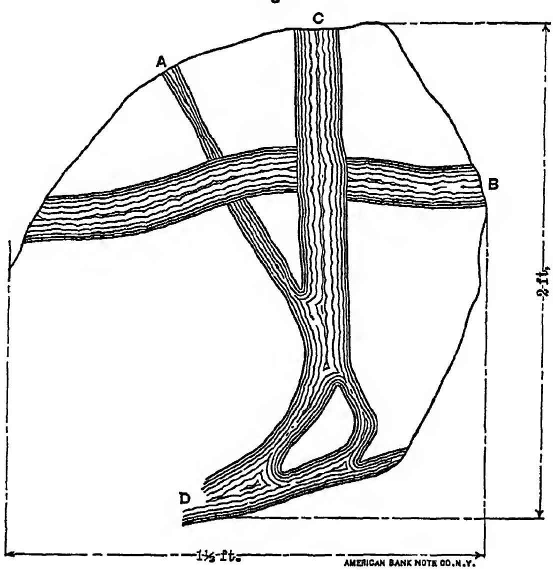


"petered" out. As a matter of fact, ore of pay-grade continued for 30 feet beyond IV.

Fig. 3 represents a boulder of hornblende-andesite, with quartz-veins of various relative ages, sketched on the Syndicate hill, Bodie, California, in August, 1896. It is introduced apropos of Mr. Rickard's remarks about twin-veins, with the suggestion that as subdivided lodes often resemble twin-veins, so veins of different ages often resemble splits of the same lode. A, apparently a branch of C, is cut by B, which again is faulted by C. It requires the most minute study of the boulder to determine that A is an independent vein, which probably swung around towards D on the inner edge of the curve, and that C,

the youngest member of the group, has attached itself to A, so that, but for the apparently anomalous faulting, the difference in age would not have been guessed. This illustrates in a small way what is very common in the veins of the Bodie district. Frequently a vein of 12 or 15 inches will be, in effect, robbed of the greater part of its value by the branching-off of a 2-inch high-grade feeder. The explanation of such cases is that

Fig. 3



the not-easily-recognizable high-grade feeder has all along furnished the chief value, and moreover, on the analogy of the above, that the feeder is an independent vein. This is conclusively proved in more than one instance in the Bulwer mine, where the small high-grade feeder is faulted by the comparatively large low-grade vein.

The point seems to be clearly made that "splits" or "feeders" are not necessarily to be considered as of the same age with the main vein.

T. E. Schwarz, Denver, Colo.: I have been particularly pleased with this paper, because of the accuracy of observation and description in the illustrations cited. I wish briefly to refer

to the term "ore-break," which Mr. Rickard uses, with the following explanatory foot-note:

"At Red Mountain, in Ouray county, Colo., it has been the practice to speak of the veins (the Guston, Yankee Girl and other celebrated lodes) as 'ore-breaks,' a break in the rock accompanied by ore; a term, it seems to me, much preferable to 'fissure-veins.'"

In the early days of the district, in 1884, when placed in charge of the Yankee Girl mine, then but slightly opened, and the only producing property, and in later years in the development of the National Belle, Genesee-Vanderbilt and Guston mines, I found it very difficult to adapt the term "fissure-vein" to the ore-deposits I was developing. Even with the most liberal interpretation of the term I was not satisfied to describe the ore as occurring in fissure-veins, and thus to identify the formation, in some degree, with that of well-known fissure-vein sections.

The ore-bodies of the Red Mountain district occur as chimneys or chutes, having great persistence in depth and varying greatly in dip and cross-section, but ramifying towards the surface. The outcrops of these ore-chutes occur along lines of alteration of the andesite or enclosing rock, and such lines were undoubtedly fracture-planes of greater or less extent. These fracture-planes, or belts of metamorphosed andesite, were in the early days of the district termed by me "ore-breaks," to signify the character of rock along the line of which the ore-chimneys occurred, or might be expected to occur. They were not fissure-veins, in that the ore-bodies did not confine themselves to any given plane, and in following them no definite course, either laterally or vertically, could be counted upon ahead of exploration. (See *Trans.*, xviii., 139.) The course of an ore-break I have frequently found to change 90° in a depth of several hundred feet.

The Red Mountain ore-bodies undoubtedly mark the course of ascending currents of heated mineral waters of the nature of hot springs, which deposited their mineral contents by replacement, and were naturally active along the lines of the so-called ore-breaks. Several surface-vents unite with depth, until the main central channel is generally found in an extended zone of kaolinized andesite at the intersection of two or more ore-breaks.

The illustration, Fig. 1, is a sketch of the formation existing on the 300-foot level of the Genesee-Vanderbilt mine, so far as determined by drifts and diamond drills in 1893. The areas A, B and C are of hard, fine-grained, blue-black rock, which, I believe, was determined by Mr. G. E. Kedzie to be hypersthene-andesite. The area between and at D consists of more or less

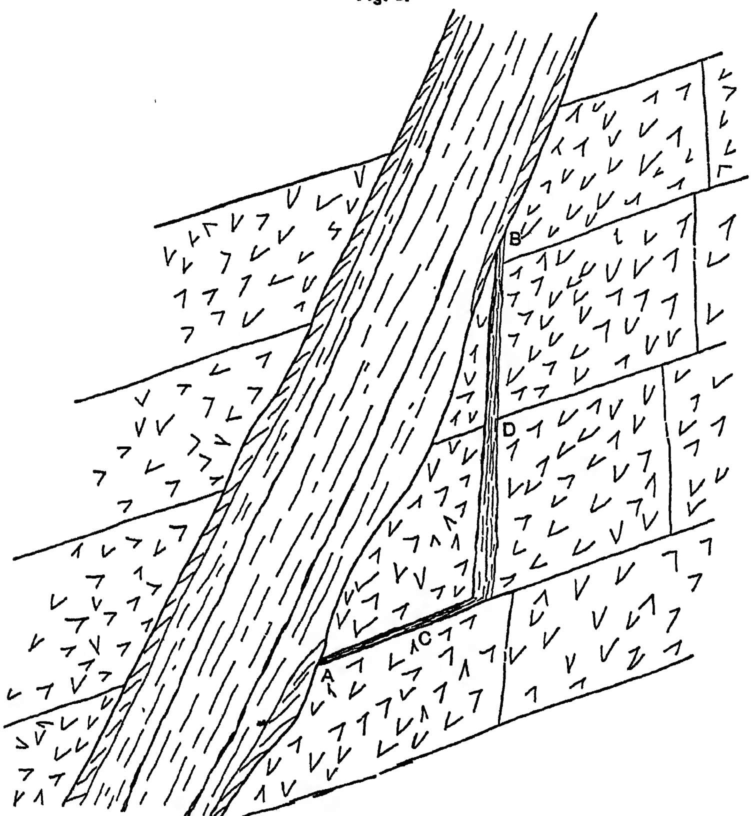
Fig. 1.

Sketch-Plan, Showing Geology on the 300-Foot Level of the Genesee-Vanderbilt Mine, Red Mountain District, Colo.

altered whitish andesite, in places changed to a soft clay or kaolin. The line of surface-outcrop of the ore-break, 450 to 500 feet above this section, is X Z, along which ore-chimneys, with their hard quartz envelopes, outcrop at V, G and S. These ore-chutes have been traced to M and O on the 300-foot

level, while on the 600-foot level they appear at N and P in a well-defined ore-break, T R. The area F H, surrounding the core of ore and quartz, O M, is a mass of soft, clayey material heavily charged with iron pyrites and containing siliceous

Fig. 2.



Cross-Section of Vein in Little Giant Mine, Warren, Idaho.

bunches, but gradually changing into a firm, light-colored andesite.

The above description will show, I think, that the formation at Red Mountain, Colo., as shown in the Yankee Girl, Genesee-Vanderbilt and other properties, is not that of fissure-veins, and that the term "ore-break" is aptly used under such conditions, and others where the expression "fissure-vein," with its widely-

understood and generally-accepted meaning, would confuse the reader.

Apropos of Mr. Brown's discussion of Mr. Rickard's paper, in which he refers to the robbing of veins in the Bodie district by small but high-grade feeders, I would introduce Fig. 2. It was sketched in August, 1896, in the Little Giant mine, Warren, Idaho. The vein is a clean-cut fissure in the granite, and averages probably 10 inches. The vein-filling is white quartz, usually carrying a pay-streak on each wall, without any gouge. The ore is very high-grade, carrying considerable coarse free gold, besides sulphides. The bedding-planes of the granite are distinct and regular. It is occasionally found that the foot-wall streak becomes barren and the hanging-wall streak low-grade between certain points, as, for example, A and B. This is accounted for by the departure at A, along the bedding-plane in the foot-wall, of a very small but very rich streak C, which returns to the main vein again at B through a nearly vertical jointing-plane in the granite, D. The streak D is much larger than C, and both are generally mined. In this case the enriching solutions found conditions more favorable for deposition of their gold and silver contents in the tight planes of the granite than in the fissure itself.

"American" Magnesite.

Discussion of the Paper of Mr. J. D. Pennock (see p. 263).

(Colorado Meeting, September, 1896.)

Frank Williams, Johnstown, Pa. (Communication to the Secretary): As the manager of the company which manufactured the "American" magnesite and the silica-brick mentioned by Mr. Pennock, I beg to say that he was quite right in placing a mark of interrogation after "American" (page 268). While the users of magnesite in this country are at a disadvantage as to cost by reason of not having any large bodies of sufficiently pure material at hand, the manufacturing concern referred to has secured an advantage as to quality by a mixture of the different foreign magnesites, which vary considerably in their composition and physical structure.

The American (?) magnesite tested was a sample of the next

to the last in a series of brick-mixtures, and consisted of three parts Austrian magnesite and one part Grecian magnesite.

It was unfortunate that Mr. Pennock was unable to obtain the variation in size at a very high temperature.

The silica-brick sample was from a mixture made in order to observe the effect of reducing the lime and increasing the alumina as bonding materials. It would have been interesting to note the change in size.

Action of Blast-Furnace Gases upon Various Iron-Ores.

Discussion of the Paper of Mr. O. O. Laudig (see p. 269).

(Colorado Meeting, September, 1896.)

F. E. Bachman, Buffalo, N. Y. (Communication to the Secretary): The investigation so fully described by Mr. Laudig was undertaken with the idea of determining if it is possible to learn by experiment, without an actual trial in the furnace, what economical results can be expected from the use of a given ore-mixture. The research was necessarily preliminary, and the results obtained are only to be depended upon so far as they relate to the samples treated.

With the exception of the ores obtained from the stock-piles of the Buffalo Furnace Company, and several samples taken at a neighboring furnace, there is no certainty that the ores represented the average shipments of any mine. For this reason the publication of the names of the ores might do producers a great injustice. The names of the ores are therefore withheld, and their identity is further veiled by leaving out of the analyses the phosphorus-contents.

The investigation was begun with the preconceived idea that the rate of reduction would be found to be dependent upon the accessibility of the particles of the ore to the action of the reducing-gas. In other words, a spongy, brown hematite or a very fine Mesabi would be the easiest reduced. I had formed a scale with No. 21 first, No. 30 second, No. 1 third, Nos. 5, 24, 25 and the remaining Mesabis fourth, etc , which is sufficient to show how much actual results differ from a judgment based on physical characteristics. The ore expected to be most easily reduced proved to be 25th in the list, while the one placed

second was by far the most easily reduced of the lot. These two ores are from the same formation, both gossans, and to all appearances identical, except in richness.

Classifying the ores, we have the following tables:

Soft Hematites.

No.	Per cent. of original oxygen lost.	Per cent. of carbon deposited.	Carbon deposited per unit of iron.
8,	26.76	23.60	.3918
14,	25.13	35.13	.5402
17,	16.99	12.32	.1985
19,	16.06	10.86	.1671
22,	15.95	12.60	.2761
18,	15.67	4.48	.0751
12,	13.07	4.64	.0728
9,	11.89	6.96	.1151
Average,	17.94	13.82	.2296

Hard Hematites.

26 *b*,	24.36	12.88	.1898
28,	13.22	2.16	.0324
Average,	18.79	7.52	.1111

Semi-Hard Hematite.

23,	6.26	3.24	.0489

Hard Specular.

33,	24.84	16.88	.2485

Magnetites.

27,	0.00	0.00	.0000
29,	0.00	0.10	.0017
Average,	0.00	0.05	.0008

Mesabis.

1,	29.37	17.78	.2763
3,	26.69	22.08	.3568
15,	25.93	36.40	.5618
2,	17.83	10.20	.1655
Average,	24.95	21.61	.3401

Blue Hematites.

11,	18.35	4.66	.0708
4,	17.40	4.72	.0755
10,	15.80	1.98	.0355
13,	15.00	2.50	.0379
6,	13.58	3.10	.0504
5,	11.33	1.56	.0239
Average,	15.24	3.08	.0501

Brown Hematites.

No.	Per cent. of original oxygen lost.	Per cent. of carbon deposited.	Carbon deposited per unit of iron.
30,	50.04	14.30	.2420
24,	25.78	24.92	.4391
25,	23.98	11.56	.2582
21,	11 98	.98	.0208
Average,	27.95	12.94	.2400

Cinders and Scales.

No.	Per cent. of original oxygen lost.	Per cent. of carbon deposited.	Carbon deposited per unit of iron.
7,	.09	.09	.0014
20,	4.07	.62	.0081
31,	.00	.08	.0017
32,	4.56	.74	.0129
Average,	2.68	.38	.0063

No. 16 is not classified, being from the Mesabi district, but having none of the characteristics of the ores known as Mesabis.

In examining these tables there is found to be nothing uniform in the reduction of the ores, every class showing great variation. Moreover, reduction and carbon-deposition bear but little relation to each other, except that where there is a large carbon-deposition there is a large amount of oxygen lost.

The only uniformity seems to be in carbon-deposition in the Mesabis and soft blue hematites—the former depositing far the larger amount, while the latter, as a class, deposit the smallest amount. If, on more thorough investigation, this is found to be a universal rule, it may prove to be of great value in making an ore-mixture. Experience has already proved certain of the blue ores to be exceptionally well-suited for mixture with Mesabi ores, preventing to a great extent the troubles arising from use of the latter.

That the results obtained bear some relation to furnace-work is shown by actual experience. For example: A furnace manager using $\frac{11}{12}$ of No. 27 and $\frac{1}{12}$ of No. 28 with Connellsville coke reports 2 tons of coke per ton of mill-iron. Another, using a mixture of Nos. 10, 15, 22 and a soft hematite not in this list, makes Bessemer iron regularly with 1700 pounds of Connellsville coke. The writer replaced $\frac{1}{3}$ of No. 19 with $\frac{1}{3}$ of No. 9, with an increase in fuel-consumption of 150 pounds per ton of iron. I have used a mixture of $\frac{2}{8}$ No. 30, $\frac{2}{8}$ No. 25, $\frac{3}{8}$ No. 24 and $\frac{1}{8}$ of an unlisted ore, and made open-mill iron with 1856 pounds of Pocahontas coke—the mixture yielding 48 per cent. through the furnace. With the same furnace and the

same kind of iron, using $\frac{3}{8}$ No. 24, $\frac{4}{8}$ No. 25 and $\frac{1}{8}$ No. 30, 2150 pounds of Pocahontas coke per ton were required. While using $\frac{4}{8}$ No. 25, $\frac{2}{8}$ No. 24 and $\frac{2}{8}$ No. 21, the fuel-consumption on foundry-iron averaged 2650 pounds Pocahontas coke, the mixture yielding 42 per cent.

In the first case cited, we have a mixture rather "too dry" for free furnace-working—even on low-silicon mill-iron with the heavy coke-consumption—which shows 1.02 per cent. of original oxygen lost and .18 per cent. of carbon deposited. In the second case we have a mixture showing very easy reduction and relatively high carbon-deposition, and from its exceedingly good work it is more than probable that the unlisted ore in this mixture is one very easily reduced. In the third case (replacement of $\frac{1}{8}$ No. 19 with $\frac{1}{8}$ No. 9) the iron in the mixture was reduced 0.40 per cent., the original oxygen lost 1.40 per cent. and carbon deposited 1.30 per cent., other things remaining the same. The fourth, fifth and sixth cases are not exactly comparable; but assuming that the $\frac{1}{8}$ of the mixture in the fourth case was as easily reduced as the most refractory of the other three ores, we have a mixture losing 31.26 per cent. of original oxygen and depositing 17.25 per cent. of carbon; and in the fifth case, one losing 28 per cent. of original oxygen and depositing 17.16 per cent. of carbon. In the sixth case, we have 21.30 per cent. of original oxygen lost and 12.25 per cent. of carbon deposited. The difference between 1856 pounds coke and 2150 pounds coke is too great to be accounted for by the difference between a 48 and a 45 per cent. mixture, as also is the difference between 1856 and 2650 pounds coke too great to be accounted for by the difference between a 48 and a 42 per cent. mixture and the difference in heat-requirements between a 1 per cent.-silicon mill and a 2.25 per cent.-silicon foundry-iron. This difference must therefore be attributed to the greater ease with which the ores give off their oxygen, and to the amounts of carbon deposited.

In all of these cases, except the first, the carbon-deposition is ample to disintegrate the hardest ore, Nos. 26 *a* and 26 *b* being reduced to powder in the experiments without a greater deposit. If, then, the ores which give off their oxygen quickest and absorb larger amounts of carbon, require the least amount of fuel per ton of iron, why is such the case? Sir Lowthian

pletely reduced by the action of either mixtures of CO and CO_2 or CO alone, at the heats of the blast-furnace. Also, that carbon-deposition ceases at a bright red heat. In the modern, rapidly-driven furnace, it is easily conceivable that a lump of ore which does not absorb carbon or lose oxygen rapidly will reach a temperature in the furnace where fusion will begin before reduction is completed. If such be the case, the furnace will go onto black cinder, carrying iron, with the well-known results. Moreover, it is reasonable to suppose that an ore which will give little of its oxygen to CO at heats lower than a red heat, will retain a greater amount of oxygen, which will only be given up by the direct union of carbon with the oxygen of the ore. As the reaction $CO + O = CO_2$ develops 5600 heat-units, and $C + O = CO$ develops 2400 heat units, the loss of heat for every unit of oxygen lost by the direct union of carbon and oxygen is apparent.

To me, however, the subject worthy of most attention is that of carbon-deposition—its cause and effect. As to the cause, I have no theory to advance; as to its effects, several. Let us consider first the economy in fuel-consumption. Assuming a furnace in which the escaping gases contain $1CO_2$ to $2.25CO$, which is an average composition, we have:

	Heat units.
1C burned to CO_2 develops	8,000
2.25C burned to CO develops	5,400
3.25 units C develops	13,400
1 unit C develops	4,123

Now let us trace 2 units of carbon, which are first burned at the tuyeres to CO, then decomposed to $CO_2 + C$, and the carbon unit deposited again burned, and finally passing off in gas of the composition of $1CO_2$ to $2.25CO$:

	Heat units.	Heat units.
2C burned to 2CO develops		4,800
$2CO = 1CO_2 + C$		
1CO burned to CO_2 develops	5,600	
1CO reduced to C absorbs	2,400	3,200
1C burned to ($1CO_2$ to $2.25CO$) yields		4,123
2C develops		12,123
1C "		6,061

Thus developing per unit of carbon 50 per cent. more heat than in the former case.

Mechanically, the first action of deposited carbon is to disintegrate the ore. This is of the greatest importance in opening up the pores for the action of CO. For this reason any ore depositing carbon freely will be easily reduced.

While it is not necessary for an ore to deposit carbon, to be easily reduced, in cases where ores are easily reduced without carbon-deposition, an examination, if sufficiently powerful microscopes were obtainable, would very likely show the molecules to be more widely separated than where they gave up oxygen less freely to CO. The carbon deposit being of the nature of lamp-black, is of necessity more easily acted upon by CO_2 at high heats. It is therefore probable that all the CO_2 from the limestone attacks this carbon-deposit and is converted to CO, any remaining carbon being carried down to that zone of the furnace where the last traces of oxygen are given up by the ore, there directly combining with that oxygen. It is rather difficult to imagine this reaction taking place where there is no carbon-deposit, but it apparently takes place, if there are any ores which deposit no carbon under the conditions existing in the furnace.

Carbon-deposition seems to have little to do with the carbon-contents of the pig-iron. The average composition of foundry-iron made from $\frac{4}{12}$ No. 9, $\frac{3}{12}$ No. 17, $\frac{2}{12}$ No. 10, $\frac{2}{12}$ No. 26 *b* and $\frac{1}{12}$ No. 22 is C 3.47, Si 2.49, Mn 0.58 and S 0.027 per cent. This mixture deposited 0.1485 carbon per unit of iron. Foundry-iron made from $\frac{4}{12}$ No. 9, $\frac{4}{12}$ No. 17, $\frac{2}{12}$ No. 10, $\frac{1}{12}$ No. 26 *b* and $\frac{1}{12}$ No. 22 analyzed as follows: C 3.97, Si 3.17, Mn 0.79, S 0.047 per cent. This mixture deposited 0.1493 carbon per unit of iron. A mixture of $\frac{2}{12}$ mixed ore, $\frac{2}{12}$ No. 17, $\frac{6}{12}$ No. 10, $\frac{1}{12}$ No. 26 *b* and $\frac{1}{12}$ No. 22, contained C 3.65, Si 1.58, Mn 0.85 and S 0.018 per cent. The ores deposited 0.1144 carbon per unit of iron. In this mixture the $\frac{2}{12}$ of mixed ore was made up of the ores in the first mixture in about the proportions given. We have a difference of but 0.08 per cent. of carbon deposited by the ores in the first two cases, with a difference of 0.50 per cent. carbon in the metal, while between the first and third there is a difference of 3.41 per cent. of carbon deposited, with a variation of 0.18 per cent. in the carbon-

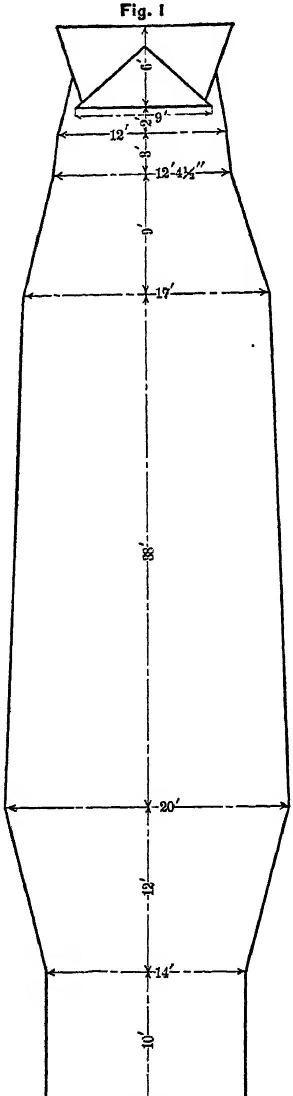
contents of the pig; the ores depositing the smaller amount of carbon giving the larger amount in the iron—a result partially explained by the higher manganese in the second and third cases.

As shown before, the average amount of carbon deposited by the soft red hematites from the old ranges is 13.82 per cent. of the original weight of the ore, the extremes being 35.13 and 4.48 per cent.; hard hematites average 7.52 (extremes, 12.88 and 2.16); blue ores average 3.08 (extremes, 4.72 and 1.56); brown ores average 12.94 (extremes, 24.92 and 0.98) per cent.; magnetites give practically no deposit; Mesabis average 21.61 (extremes, 36.40 and 10.20); cinders and scales average 0.38 (extremes, 0.74 and 0.08) per cent. This shows that the Mesabis deposit almost twice as much carbon as any other class of ores treated. If the samples treated represent the average of this class of ores, this fact must have a bearing upon their use; and I think it explains the cause of some of the trouble their use has occasioned. The ores of this class were of double their original bulk after burning off the carbon, the carbon and ore together making 4 or 5 times the bulk of the original sample. As we found the gas-pressure to increase from 2.35 ounces to 10 ounces in descending the furnace 10 feet, we can readily see the effect of using an ore as fine as No. 1, 2, 3 or 15. We have approximately, in this case, two volumes of coke and stone to one volume of ore. If the ore-charge is lumpy or even of the finer soft red ores, the charge will be, to a certain extent, stratified; but if it is very fine and sandy, the ore will fill all the openings between the lumps of coke and stone. The ore then deposits carbon, and is expanded to 4 or 5 times its original volume, giving in the charge 1 volume of coke and stone to 2½ volumes of ore and carbon; or, in other words, the volume of the charge is increased from 1½ times the volume of the coke and stone to 3½ times their volume in a descent of 10 feet, while the area of the furnace is not increased in proportion. The effect of the packing is that the gas channels passages through the stock, the breaking through of these channels or chimneys being indicated by puffs of black smoke with an excess of gas for a few seconds, a shower of flue-dirt and a relief of the pressure in the upper parts of the furnace.

That these puffs are not necessarily slips, is easily shown by

gauging the furnace. At times, all the gas passes through these chimneys, the charges descending until they reach a temperature where reduction takes place by the direct union of solid carbon with the oxygen of the ore, causing a sudden and unaccountable cooling of the furnace. Again, the stock packs so tightly that the gas cannot form a channel, in which case it takes but a few seconds to raise the pressure in the whole furnace to such an extent that when it becomes sufficient to force a passage, the down-comer is not large enough to accommodate the excessive amount of gas, and the bell and hopper are laid on the platform or thrown over the side of the furnace in the too familiar top-explosion, so common when Mesabi ores were first introduced.

Fig. 1



AMERICAN BANK NOTE CO. N.Y.

Furnace-Lines Suggested for Use with Mesabi Ores.

If this theory is correct (and there are a great many facts to confirm it, such as the lack of top-explosions in furnaces using larger amounts of cinder with Mesabi ores, and the fact that ores Nos. 10 and 11, which are rather lumpy, deposit small amounts of carbon and reduce readily, have been found to work admirably with Mesabi ores), what is a desirable ore-mixture?

First, one which has a sufficient amount of slag-making material to give a regular working furnace, and take up the sulphur in the charge without giving over 2 per cent. of sulphur in the cinder.

Secondly, one in which all the ores lose their oxygen rapidly and uniformly at the lowest pos-

sible temperature; for if one loses its oxygen slowly, either an excess of fuel will be required or the furnace must be driven slower to accommodate that ore. In other words, a mixture in which each ore loses 25 per cent. of its oxygen in a given time can be driven faster than if one ore loses 49 per cent. of its oxygen in the same time, and the other loses 1 per cent.

Thirdly, one in which there is a sufficient carbon-deposit by each ore to (1) thoroughly disintegrate it, (2) convert all the CO_2 of the flux to CO, and (3) remove all the oxygen from the ore that is not removed by combination with CO, and as much more as possible without causing top-explosions and irregular working.

Where it becomes necessary to use ores depositing too much carbon, the troubles from their use can probably be prevented by mixing these with ores depositing small amounts of carbon, especially if they are lumpy. What amount of carbon-deposit may be allowed with safety will depend to some extent on the fineness of the ore originally, as lumpy ores will not fill the crevices between the coke and stone as readily as fine ores, and therefore Nos. 1 and 3 will be more likely to give trouble than Nos. 8 and 14. If, after further experience and research, these theories for the causes of the difficulties in working Mesabi ores prove to be correct, a change from the ordinary furnace-lines may possibly prevent the trouble arising from their use, and a furnace constructed to prevent packing of the stock below the stock-line is the natural solution. For a 20- by 80-foot furnace I would suggest, as worthy of a trial, the lines shown in Fig. 1.

Finally, the investigation made by Mr. Laudig was, as stated by him, largely preliminary. To arrive at more definite results the investigation must be made on ores that are accurate average samples, and more nearly with actual furnace conditions. The pressures of the gas should have been higher, more nearly approaching the 10 ounces found in the furnace, and the temperature should have been gradually raised from that of the tunnel head gas to a red heat, the time of the action being made the same as that of the ore in passing through these temperatures in the furnace. The amount of gas should be the same as passes over the same weight of ore in the same time in the furnace. If these conditions are secured, the results cannot fail to be of the greatest value.

Ralph H. Sweetser, Everett, Pa. (Communication to the Secretary): In his discussion of Mr. Laudig's interesting paper, Mr. Bachman speaks of the effect of the packing of the stock near the top of the furnace due to the carbon deposition. He says (p. 1067):

> "The effect of the packing is that the gas channels passages through the stock, the breaking through of these channels or chimneys being indicated by puffs of black smoke, with an excess of gas for a few seconds, a shower of flue-dirt, and a relief of the pressure in the upper parts of the furnace."

These puffs, sometimes lasting two or three minutes, have been frequently noticed in furnace "A" of the Maryland Steel Company, and especially while oyster-shells were used as a flux. During the more violent puffs oyster-shells and pieces of coke would be thrown against the bell and into the down-comer, and small pieces would come out of the bleeder; dense black smoke would appear at the top of the chimney; and the pressure of the gas in the mains would sometimes lift the explosion-doors. There would be neither explosion nor any slip. Several times after such puffs the furnace was gauged; but seldom had it settled, and then not more than half a charge.

During one of these puffs a sample of the waste gas was taken from the down-comer (see Analysis I., below). While the sample was being taken, the top-temperature suddenly rose from 680° to 890° F., and this time the furnace slipped from "full" to "half a charge down." The mixture was wholly of foreign ores, $\frac{1}{3}$ Mokta, $\frac{1}{3}$ Elba lavato, $\frac{1}{3}$ Tafna. Mokta is hard and lumpy; Elba lavato is a washed granular ore; Tafna is a soft, fine hematite. The flux was wholly oyster-shells.

The ratio of CO to CO_2 in the gas-sample was 2.08, the lowest ratio that had been recorded for that furnace up to that time (May 2, 1896). On the supposition that this decreased ratio and sudden outburst of gas were due chiefly to the rapid decomposition of the oyster-shells at this temperature (680°–890° F.), a gas-sample was taken when the furnace-top had quieted down and the gas-pressure was normal.

Analysis II. was made on this sample.*

* Since the above analyses were made the ratio of CO to CO_2 has been much lower, and has gone as low as $\frac{CO}{CO_2} = 1.32$.

Analysis of Furnace-Gas From Down-Comer.

(Results Given in Per Cent. by Volume).

	CO_2	CO	H	CH_4	N, etc.	$\frac{CO}{CO_2}$	Blast-Pressure lbs.
I., ...	12.4	25.8	2.3	1.0	58.5	2.08	14
II., ...	10.2	29.0	1.2	1.2	58.4	2.84	13.5

The temperature of the top of the furnace was about 890° F. when the first sample was taken. Should the larger amount of CO_2 in the waste-gas at that moment, and the violent outburst be ascribed to the sudden liberation of CO_2 from the oyster-shells, or to the causes described by Messrs. Laudig and Bachman ?

Electricity in Mining.*

Discussion of the Papers of Mr. Robert G. Brown (see p. 319) and Mr. Irving Hale (see p. 402).

(Colorado Meeting, September, 1896.)

Mr. Brown: That part of the Standard Company's plant which has been added since the publication of Mr. Leggett's paper, and is described in mine, renders the single-phase, alternating system, which Mr. Hale has called "unelastic," much more elastic than it had been. The original plant at Bodie was substantially the same as the Telluride plant mentioned in his paper. The machines came from the same shops, and were practically the same; but while the Telluride Company has changed completely to a polyphase system, the Standard Company has retained its original plant, but has belted on a direct-current generator at the receiving end, thereby converting the water-power into electricity, electricity into power, the power into electricity, and finally again electricity into power for use in the mines. This is a roundabout course, to be sure, but it shows that there are possibilities even in the out-of-date single-phase system. The only point perhaps, that deserves particular notice with respect to the hoist is the fact that the rheostat, or starting-resistance, was not built originally

* Secretary's Note.—The stenographic report of this discussion has been corrected, with some additions, by the participants, and those parts of the remarks of Messrs. Hale and Brown which consisted merely in an oral résumé of their printed papers have been omitted.

of sufficient carrying-capacity. Engineers will do well to bear in mind that this will be the danger. In two rheostats on hoists of that kind the fault has been the same—namely, that in constant running, with frequent starts and stops, the rheostat carrying a great part of the current at times, the heating is tremendous. In Bodie we have had many times to stop hoisting in order to allow the rheostat to cool off, for fear of warping or cracking the castings.

In addition to the detailed account given in my paper, which goes into many points of construction and cost, I should like to make one or two remarks. It is good for all of us to see where the faults lie in any established plant, thereby giving us the opportunity to avoid them in the future, and I am frank in admitting that in this plant of the Standard Company there is one fault in the water part of the work, and that is in the failure to establish a suitable air-cushion at the end of the pipe-line. A person walking along the pipe-line and putting his hand upon it, can feel at times a quick throb. That is due to the sudden partial stoppage of the flow of water in the pipe-line, caused by throwing off the motor in the mine 12 miles away, which immediately causes the governor to act, closes the gate partially and makes the water slow up in the pipe. This is a severe strain upon the pipe, and, although we have had as yet no trouble in the matter, it is not pleasant to feel that there is a constant pounding of that kind going on as many times a day as the hoist is thrown off or on.

Another point of particular interest to us has been the trouble with lightning. In the early starting of the plant a delay of some months was caused by continual trouble from this source. After that a set of Wurtz lightning arresters was installed, which are described in my paper.

I would add that it is exceedingly doubtful whether there are any static accumulations on the line. Many tests were made by Mr. Davis of the Westinghouse Company with the pith-ball electrometer, and he could find no evidence at all of dammed-up or static electricity on the line.

From the description given in my paper it is very clear that, as far as we are concerned, the safeguard against lightning is not satisfactory.

Mr. Hale's interesting paper leads me to wish earnestly that

somebody would do for the Sierra region in California what he has done for the Rocky Mountain region. There is a great deal of very interesting practice going on there at present. A great many very interesting installations have been made; but the descriptions given of them have been mostly imperfect and unsatisfactory.

So far as I can find, we have not accurate data at hand for comparing the advantages of electricity and compressed air. I fully appreciate the advantages of electricity. If it were not for the Standard power-plant, the Standard Co. would be scarcely running to-day; but, in view of the extremely interesting installation of compressed air at the North Star mine, at Grass Valley, California, where they compress air directly by the use of their water-power on an 18-foot Pelton wheel, and transmit the power a short distance, and then use the air reheated to a temperature of 350 to 400 degrees, thereby securing a great amount of efficiency at a low expenditure of heat, I say that, in view of this, it does not seem to me that we have as yet definite data for determining the question of relative economy between the two methods of transmission. Of course the simple transmission-cost per horse-power would be a controlling consideration in cases of transmission for many miles; but when it is a question of concentrating the power-plant at one place, or carrying power from water at the foot of a hill up to the top of it, a mile or so, it is by no means a foregone conclusion that electricity does the cheaper or more satisfactory work. There has been more than one attempt to solve this question definitely; but those that I have seen have been all in some way unfair in the study of the question, which has been stated apparently with the idea of proving one side or the other, instead of weighing the facts and determining which side should have the preference.

In conclusion, I should like to ask Mr. Hale what is the system of lightning-arresters in general use in the Rocky Mountain region and what success it gives, and I should like to ask in that connection whether the use of overhead barbed wire, which has been spoken of as having been used on several lines, has been shown to be of advantage, and also whether there has been any attempt to store excess of power at times by means of storage-plants?

H. S. Chamberlain, Chattanooga, Tenn.: I would like to ask Mr. Brown whether the trouble with lightning is greater at high altitudes? We do not hear very much about it at lower altitudes in the East.

Mr. Brown: Recent experiments tend to show that the tension of atmospheric electricity increases with altitude. It is this tension, or high potential, that makes atmospheric electricity dangerous to the insulation of electrical machinery.

The electric energy certainly seems to be more intense at high altitudes generally. We do not have the fierce, bright flashes of lightning that we often see in damp countries; but we have very intense, sharp thunder-storms, which, perhaps, give three or four sharp claps with very brilliant flashes, and then the storm passes.

Mr. Hale: Since my paper was first put in type I have added to the summary table appended to it a few machines since installed, and have inserted the *approximate* capacity of the Great Falls plant (2800 k.w.), accurate data for which were not obtainable and were not included in the totals, thus increasing the total generator capacity from 7988 to 10,964 k.w. and showing 135 motors aggregating 5201 horse-power, distributed as follows: On hoists, 1169; pumps, 681; blowers, 112; air-compressors, 403; electric percussion-drills, 20; diamond-drills, 17; locomotives, 165; placer-mining machines, 208; mills, 2155; machine-shops, 61; coal-screens, 10; unknown, 200 horse-power.*

The paper of Mr. Brown and the previous paper by Mr. Leggett are certainly of great interest and value, in that they describe a particular plant and go into detail concerning a great many practical points, which it is impossible to do in a single paper attempting to cover all the establishments in a large region. This plant at Bodie is very interesting, as it illustrates the evolution of electric power and the advantages and disadvantages of the different systems that have been used at different times. It illustrates also the unpleasant dilemma in which the user of electric power is sometimes placed in choosing a system, because the electrical companies are not accommodating enough to get up an absolutely perfect system at the very outset. When they put in this plant—I think it was 1892—the

* Secretary's Note.—The Table opposite page 416 contains the additional data here mentioned.

direct-current system was out of the question, because they were obliged to transmit about 12 miles, and the cost of copper at 1000 volts (which is about the highest practicable voltage for direct-current), with 10 per cent. loss in line, would have been something like $800 per horse-power, or for the 120 horse-power transmitted about $100,000 for copper alone, or nearly three times the cost of the whole plant as actually installed. That, of course, made the direct-current system impracticable.

Mr. Leggett mentions that they also had presented to them the direct-current system using several generators in series, and also several motors in series, thus getting a high voltage; but that system is evidently complicated and unsuitable where power is used for general purposes. At that time the multi-phase system had not been brought out, so they were obliged to fall back on the single-phase alternating system which gave high voltage and permitted long-distance transmission, although it had the disadvantage, as explained in my paper, that these single-phase alternating motors are not self-starting nor capable of speed-regulation, and it requires about 5 minutes to get the motor started and up to speed, so that the current can be thrown on. This system is evidently unsuitable for hoisting and pumping, or general power-purposes, but it does very well for the case in hand, which is to run a mill steadily 24 hours a day, and every day in the week. The increase in the plant described in Mr. Brown's paper illustrates, however, the shortcomings of that system, because, as soon as they wanted to apply the power to hoisting and pumping, it was necessary to put in a direct-current generator, run by an alternating-current motor, in order to supply the direct-current motors on the hoist and pump. This involved the cost of an additional motor and generator, the complication of the plant by two more machines to look after, and a loss of about 20 per cent. in power in these two machines, all of which could have been avoided if the motors could have been run direct from the main circuit. If the electrical people had had the forethought to bring out the multi-phase system three or four years sooner, all this would have been saved.

Mr. Brown mentioned the point that the rheostat on his hoist heats badly, that is, when it is used continuously. I simply

want to call attention to the fact that this is not an inherent defect in the electric hoist. The trouble is that his rheostat is not large enough; has not sufficient capacity for that particular service. If the rheostat is to be used for controlling the current, and not merely for starting, and is to be in circuit most of the time, it must have greater capacity. This is illustrated by the odor from the rheostat in circuit with this magnetic separator* during the early part of the evening; it is not large enough for the current that has to go through it, and the heat softens the insulation.

Mr. Brown inquired about the lightning-arresters used in this region. This is one of the most important things to be considered, especially in mountainous countries, where lightning is always very severe.

It may be said that there are three types of lightning-arresters now in general use, each of them perhaps possessing some advantages over the others. The one which he describes consists of a number of cylinders of non-arcing metal, placed parallel and close together, with little gaps between, so that the lightning jumps across; and this metal being non-arcing, as it is called (that is, it does not establish an arc or bridge under the action of the current), the dynamo-current does not follow the lightning-discharge. In another type of arrester the lightning jumps across a small gap, and there is a strong magnet placed with the lines of magnetic force across that gap, and these magnetic lines have the effect of blowing out the arc or stopping it. This is called a "magnetic blow-out" arrester, and has been very widely used. In a new arrester, just brought out, which gives promise of being better than either, the gaps are between large metal disks of sufficient surface to radiate the heat so rapidly as to dissipate the arc in that manner. This makes a very simple and cheap arrester, and is, apparently, very efficacious.

In this district the subject of arresters has received a great deal of attention, and many types have been used with more or less success.

Perhaps one of the best examples of power-plants of this kind is the Silver Lake Mines plant at Silverton, where the

* A Wetherill magnetic separator, exhibited by Mr. Nitze. See page 357.

power is transmitted to an altitude of 12,300 feet, which is about 3000 feet higher than at Bodie. Lightning, as you know, is extremely severe during the summer in that part of the country, and yet, during the past season, as I am informed by Mr. Stoiber, the owner of the plant, they have had no trouble from lightning, and, so far as I know, were not shut down for an instant on that account. It may be said that, with the present lightning-arresters, while lightning is a thing that cannot be absolutely guaranteed against—it is very tricky—still we may feel that electric machinery has received reasonable and almost perfect protection against this danger.

In regard to the comparison in economy between electricity and compressed air, I fully agree with Mr. Brown that there is a great lack of really accurate data on which to base a judgment. A case in point is the plant just installed for the Colorado Fuel and Iron Co., at their Rouse mine, for underground pumping and other uses, but principally for pumping. This plant replaces compressed-air pumps. I have been informed by the engineer who supervised the plant that a few rough tests which he has made indicate an astonishing superiority of electricity over compressed air; the difference is so great that I would hardly care to state it as authoritative, and, in fact, he did not give it in that manner. He intends to make more accurate tests on which to base a better judgment. It should be said, also, in this connection, that the dynamo is run by an economical modern compound engine, while the air-compressors were run by less economical engines, and there were a good many other conditions in favor of electricity in this particular comparison. Still, it is my opinion, although I cannot at this moment produce accurate data to show it, that electric power is very much more efficient, and delivers a much larger percentage of the initial power than compressed air, and that it is also cheaper, provided the transmission is for any considerable distance. If I remember correctly, the tests made on the Paris system of distribution by compressed air showed a total efficiency of something like 33 per cent.—this is given simply from memory—which, of course, is very much below that obtainable by electricity, which runs up to 60 or 70 per cent., and, in some cases, considerably higher.

With regard to Mr. Brown's inquiry concerning the storage-

battery, I would say that this appliance is not used anywhere in this part of the country in connection with power-plants. The great objection to the storage-battery at present is that it is very heavy, and very expensive for a given amount of power stored; and its deterioration is also comparatively rapid. It is used with good effect in lighting-plants in many places, for assisting the dynamos during the heavy load in the early evening, or carrying the light load during the day when the plant is not running; but to store any great amount of power to tide over several days, or a week, or a month, or a period of dry weather, the cost would be so enormous as to be absolutely prohibitive. No doubt, improvements will be made in the storage-battery, and it will eventually assume a wide field in electric operations; but I am not inclined to look very hopefully on the use of the storage-battery in connection with power-transmission plants pure and simple, for the reasons stated. Moreover, it cannot be used with the alternating system, which is now employed.almost exclusively in long-distance transmission.

In answer to Mr. Brown's inquiry about the use of barbed wire as a lightning-protection (that is, a barbed wire running along the tops of the poles and grounded at intervals of a few poles by a wire running down into wet soil), that method is used in a number of plants in this region, such as the Silver Lake Mines plant at Silverton, the plant at Telluride, and, I believe, the Virginius mine at Ouray, and some others. It is difficult to tell just how much good it really does, or whether it does any good or not, because you cannot see the lightning when it strikes the barbed wire and goes down to ground as you can when it passes through the lightning-arrester. Still the general opinion seems to be that the barbed wire does give considerable protection, and that it is worth while to put it up as an auxiliary to lightning-arresters.

D. W. Brunton, Aspen, Colo.: In Aspen we have so far experienced no trouble whatever from lightning. Our wires pass through the Cowenhoven tunnel, underground, for a distance of 1½ miles, and are in some places okonite-covered cables; in others, where the timbering consists of what are known among miners as square sets, we use ordinary bare wires secured in the corners on glass insulators.

Regarding the question which has been raised here as to the

comparative efficiency of compressed air and electricity, I would say that we ought to hear on this subject from Mr. Kebler, who has recently taken out a large compressed-air plant and replaced it with electricity.

In my own experience, when we ran a 60-horse-power plant with compressed air in the Della S., the numerous objections to compressed air compelled us to take it out and replace it with electricity; since which time all of our hoists on which air-engines were employed, have been replaced with electric motors, which have proved entirely satisfactory.

In the Free Silver (the plant mentioned by Mr. Hale yesterday as the largest electrical hoisting-plant in the world), we use a direct current of 500 volts, and have so far had no trouble whatever. At first, the trouble complained of by Mr. Brown in the rheostat showed itself very plainly, and the small rheostat had to be replaced with a much larger one, since which time heating has entirely disappeared, and we can hoist at any rate of speed desired without the slightest trouble from overheating.

The hoist is overbalanced, and experiments made to determine the relative efficiency of the unbalanced or direct hoisting-system and the overbalance, showed an increase in the efficiency of 66 per cent. in favor of overbalancing.

In the Della S., we have recently been compelled, in order to cut down our electric rental, to adopt overbalancing, and have attempted a modification of the overbalance system by simply winding a cable around the hoisting-drum in the opposite direction from the cable attached to the cage. Owing to the fact that the wire cable had to be carried over two sheaves and bent twice, our efficiency there was not so great as at the Free Silver. The saving in overbalancing at the Della S. amounted to exactly 50 per cent. By overbalancing, I mean that we have counter-balanced the weight of the cage and the car, and in addition to that, have added weight equal to one-half of the live load to be lifted. For instance, if the car holds 3000 pounds of ore, then our overbalance would equal the weight of the cage, car, and 1500 pounds, so that the motor, instead of using a sufficient amount of current to lift the entire load during the actual time of hoisting, uses one-half that current, but for double the time.

In the Alta Argent plant, described by Mr. Hale yesterday, we have exceptional opportunities to use electricity under the most favorable conditions. The plant is situated about 1¾ miles underground and about 1000 feet below the surface, and is at the head of what we term an inclined shaft. The cage runs on iron wheels, and, instead of carrying the ropes up over the sheaves and down to the level, the shaft is extended above the level of the station a distance of 40 to 45 feet, and the hoisting-plant is put at the head of that opening, so that the cables pass from the drum directly down the shaft, and the electric wires are carried down the sides of the raise to the very collar of the shaft. The motor-room above has a locked door, and the engineer is entirely dispensed with. The cage-tender, standing at the collar of the shaft, operates the hoist, as all of the levers are located practically at his hand in a position where he can look down the shaft if necessary. This plant has been running for a year and a half without a single break-down or the expenditure of one cent for repairs; and the saving of three engineers' wages during the 24 hours, at $3.25 per shift, makes a total of $9.75 per day. The miners working in the bottom of the shaft at first objected to this arrangement; but they are now the most satisfied lot of men that I know of. We have sunk nearly 300 feet since this plant was installed, making the total depth from the surface about 1300 feet, without a single accident.

Perhaps it would be well for me to explain the words "inclined shaft." The rocks on Smuggler mountain dip at an angle of about 55 degrees, and some years ago, when it became necessary to follow the ore-zone below the level of the deep tunnels by which the mines there are operated, it became a grave question whether we should sink a vertical shaft and drift over to the ore-body or run the ordinary incline, following the ore-zone. Every miner knows the objections to the ordinary incline. It is expensive, dangerous, requires a large station at each level, and accidents are continually happening in it, under even the best of management. On the other hand, if we sink a vertical shaft we encounter a water-bearing stratum of rock about 40 feet below the ore-zone, which is quite persistent throughout Smuggler mountain; and, as each successive station was reached in sinking the shaft, drifts would not only

have to be run further and further to strike the ore-body, but each one of them would pass through this water-bearing stratum and receive the full effect of the immense volume of water and quicksand which it carries.

Under these circumstances the idea occurred to me that if some movement should occur in the earth's crust whereby a vertical shaft should be thrown out of the perpendicular, it would not cease to be a shaft because it was no longer perpendicular. Acting on this suggestion, we have planned what has been called an inclined shaft, which is simply an ordinary shaft tipped on its edge, instead of laid over on its side, as an incline would be. The advantages in sinking are very marked. There is less roof exposed, and where pumps are necessary they can be placed in the upper compartment of the shaft, and by putting them upon a heavy wooden platform it is possible to blast without the slightest danger of injuring them.

The Della S. inclined shaft has been sunk a distance of nearly 800 feet, and I do not remember that we have had a single accident. In fact, when blasting is going on, the bottom of the shaft is always a sufficient distance below the timbers to drive the rock up against the platform underlying the pump, which has proved ample protection under all circumstances; and I would recommend any one who is called upon to follow a sloping ore-body to look into the inclined-shaft arrangement before commencing operations.

Julian A. Kebler, Denver, Colo.: I have not had very much experience as to the relative efficiency of compressed air and electricity. We have recently put in an electric plant at our Rouse mine in Huerfano county. In order to understand the situation, some little explanation is required. The coal-mine was opened by a slope going down on a grade of about 5 per cent. When we opened the mine none of the other mines in the immediate vicinity had much water, and the slack, or fine coal, was not salable and was thrown away. We, therefore, did not care how much fuel we used. We encountered a small amount of water, put in an ordinary straight-line pump and operated with steam, carrying it some 2000 feet. At this point it was so wasteful that we put in an air-compressor. Driving the slope still farther, we encountered more water, so we put in a second, and finally a third com-

pressor. Thus, at the time the electric plant was put in, we were pumping about 600 gallons a minute through 4000 feet of pipe, having a vertical lift of about 100 feet. The compressors were all Norwalk, 22 by 24. Our air-supply pipes were not nearly large enough, as we were carrying the air from three compressors through two 4-inch pipes. It was not possible to heat the air before using, as the pumps were 4000 feet from the mouth of the mine. The pumps used were the ordinary straight-line steam-pumps of Worthington, Cameron and Smith-Vaile patterns.

Last fall we encountered another flow of water, and something had to be done. We either had to put in more air-compressors and larger air-pipes, or an electric plant; and we finally decided to try electricity. For the past two or three years our slack has been salable, and therefore we wanted to put in an economical pumping-plant; and at the same time we decided to apply electricity to our other machinery around the works. We, therefore, put in a compound-condensing high-speed engine, coupled direct to a dynamo, and put in motors to run not only our pumps but also fans, screens, and machine-shop. Therefore, in our case, we can only compare the use of compressed air under the most adverse circumstances with electricity under nearly the most favorable. While we have not had very thorough tests, the result has certainly been remarkable.

The plant has been running one and one-half months, and we are doing the same amount of work with 5 tons of coal that with compressed air took 35 or 40 tons.

Regarding the efficiency of compressed air, we have probably thirty air-compressors of different makes in use at our plants, all of them simple straight-line compressors. With the best tests that we could get we have never been able to get over 40 per cent. efficiency in transmitting the air 4000 feet.

R. W. Raymond, New York City: An interesting comparison with the figures of cost given by Messrs. Hale and Brown is furnished by a paper read before the Institute in May, 1877, by Mr. N. S. Keith, entitled, "Can We Transmit Power in Large Amount by Electricity?" This paper was suggested by a statement reported to have been made by Dr. C. W. Siemens, that a continuous rod of copper, 30 miles long and 3 inches in diameter would be capable of conveying for that distance electri-

cal energy equivalent to 1000 horse-power—the source of power being a waterfall. Mr. Keith calculated the cost of an installation for this purpose (1000 volts being his figure for the required intensity) as $40,000 for the dynamo, $33,200 for the motor, and $1,411,800 for the conductor—sums which seemed to be certainly prohibitory. But the problem has been entirely altered by the newer systems of transmission, then unknown; and his calculation stands to-day as merely an accurate landmark, from which the progress of twenty years can be measured. We may even say that the question has been radically changed since Mr. Rothwell read at the Buffalo meeting of October, 1888, his paper on "The Present Status of Electric Transmission of Power,"* a paper which may still be consulted with advantage, as an impartial discussion of the relative economy of compressed air and electricity, including a prophecy of what has since been accomplished with the latter agency.

Mr. Brown's question as to the use of the storage-battery suggests a further application of electrical energy—I mean, in connection with wind-power. Thirty years ago I gave some attention to the subject of the use of wind-power for running stamp-mills in arid and unforested districts, where coal was not available, so that neither steam nor natural water-power could be economically employed. The only method then occurring to my mind for equalizing the fitful character of wind-power was the pumping of water or the raising of weights for storage of energy. My report as United States Commissioner of Mines and Mining, published in 1870, contains (p. 717) a chapter on the subject. I need hardly say that such schemes are not particularly convenient or economical of execution. But I am interested to know whether the storage of electrical energy may not promise better results. I understand that Mr. Brush, the well-known electrician, is using wind-power and storage batteries combined for a system of electric lighting at his own country-house. Is there any probability that a similar combination could be used for power?

Mr. Hale: To the running of dynamos by wind-power and the storage of the power in storage-batteries, those objections apply which I have already stated in connection with Mr.

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* *Trans.*, xvii., 555.

Brown's general question; and in addition to them, we have the serious defects of wind as a motive-power.

In the first place a wind-mill to produce a large amount of power, is an extensive and expensive institution. In the second place, the speed is very unsteady; whereas the dynamo requires a steady speed. A dynamo has recently been designed and patented, to be used especially with wind-mills, which is intended to maintain a constant voltage at varying speed, but I believe it has not been introduced to any great extent, and I am not certain whether it is a practical success or not. In any case, however, I am inclined to think that the enormous cost of the wind-plant in proportion to the power it will develop, and also the impossibility of storing any large amount of power by any means known at present, will prevent for a long time any great utilization of wind-power. Mr. Brush's plant, which has been mentioned, is merely a very large and extremely expensive toy. As I understand it, it is more of a curiosity than a utility, and Mr. Brush built it principally to show what could be done in that direction—merely as a curious application of wind-power for the generating of electricity.

P. S.—(Communication from Mr. Hale to the Secretary, January, 1897): In the discussion at the September meeting, of the relative economy of electricity and compressed air, the scarcity of accurate comparative data was commented upon, and reference was made to some rough tests made on the compressed-air and electric pumping-plants of the Colorado Fuel and Iron Company, at Rouse, Colo., Mr. Kebler, General Manager of that company, stating that with the electric plant they were doing with 5 tons of coal the same work that required 35 to 40 tons with compressed air. Mr. Searing, the consulting engineer of the plant, has since made some thorough comparative tests, which were embodied in an interesting paper published in several technical journals. The following extracts from this paper give a summary of his tests:

The test of the compressed-air system was confined to the one pump in No. 1 slope, to obtain the total efficiency between the steam-cylinders of the compressors and the water-discharge of the pump.

Average steam-pressure, ...	80 lbs.
Average air-pressure, ...	50 "

Average revolutions per minute:	
Three compressors,	64
Two compressors,	110
Average gallons discharged by pump per minute,	400
Average strokes of pump per minute,	176
Average pressure per square inch pumped against,	150 lbs.
Average indicated horse-power at steam-cylinders of compressors,	312

As the theoretical horse-power required to force 400 gallons per minute against 120 pounds is 28, it follows that the efficiency of the system between the points measured is less than 9 per cent.

If we go back of the steam cylinders to the coal-pile, we find a still more deplorable state of affairs; for to operate these three compressors requires eight boilers, consuming 35 tons of coal per day of 24 hours.

The average of numerous tests on the two small electric pumps gave the following results:

Average gallons per minute discharged,	500
Average pressure at pumps,	56.5 lbs.
Average strokes per minute, each pump,	172
Average amperes,	37
Average station-voltage,	475
Average indicated horse-power at engine,	32.86
Average horse-power at pump discharge,	16.4
Efficiency:	
Generating-unit, engine and dynamo,	71.5 per cent.
Motor, pump and line,	70 "
Total from cylinder of engine to water-discharge of pump,	50 "

As soon as the first station-pump was started and in good running order, tests were made with the following results:

Average gallons per minute discharged,	650
Average pressure at pump,	64 lbs.
Average strokes per minute,	188
Average amperes,	53.5
Average station-voltage,	550
Average indicated horse-power at engine,	53.6
Average horse-power at pump-discharge,	24
Efficiency:	
Generating-unit, engine and dynamo,	73 per cent.
Motor, pump and line,	61 "
Total from engine cylinder to pump-discharge,	44.5 "

Although the total efficiency with the small pumps is as high as was predicted, it is nevertheless true that the engine and dynamo are working at a disadvantage with a load of but 20 per cent. of their rated capacity. In the case of the large pump, the generating unit is underloaded, as is also the pump, which combine to bring the total efficiency below the estimated 50 per cent. When the mine is finally pumped out and all the pumps and motors are running, the generator and engine will have a comfortable load, and it is safe to say that the total efficiency of the pumping-plant will then be very close to 60 per cent.

Qualitatively speaking, actual results with the compressed-air and electric plants which are the subject of this article show that the work of pumping, calling for 312 horse-power and 8 boilers in the case of compressed air, could be done by electricity with but 56 horse-power and one boiler, disregarding even the use of a compound engine and a condenser.

Further, the writer has no hesitancy in saying that the whole plant of pumps and machinery formerly requiring the eight boilers, taxed to their utmost, can be operated by the present electric plant with but the one horizontal tubular boiler.

Due allowance should be made for the fact that the compressed-air plant was not as economical and perfect an installation of its class as the electric plant. The pumps were similar in general quality, the direct-acting air-cylinder of the compressed-air pump being replaced in the electric pump by a motor transmitting its power through gearing. The plants are therefore about on a par as far as the pumping-end is concerned. But the pipe of the air-plant was too small for economical transmission, while the wiring of the electric plant was properly proportioned. The Norwalk air-compressors, with compound air-cylinders, compare favorably with compressors generally used in mining work, but the steam-end was not compounded, as in the engine running the dynamo. This, however, while it would affect the relative amounts of steam and coal used to give the same indicated horse-power in the steam-cylinder, would not affect the efficiency of the system beyond this point, as measured by the ratio of the work done at the pump-discharge to the I. H. P. in steam-cylinder—which efficiency was 9 per cent. in the compressed-air plant and 50 per cent. in the electric plant, in spite of the latter being loaded to only 20 per cent. of its full capacity. At full load this efficiency is expected to reach 60 per cent. Heating the air before using would increase the efficiency, but this is usually impracticable in underground mining work. On the whole, therefore, the air-plant under consideration is, with the possible exception of the pipe, a fair sample of plants of this kind generally found in mines, and no improvement in the pipe could overcome more than a small portion of the difference in efficiency between the two systems.

During the discussion I stated from memory that tests of the Paris compressed-air power-transmission system, one of the most perfect in the world, showed an efficiency of something like 33 per cent. I have since looked up a report of Prof. Kennedy's tests at Paris, given in a paper by Mr. L. B. Atkin-

son before the British Institution of Civil Engineers, and find that he gave the efficiency as 31 per cent. It is said that by heating the air before it enters the motor this efficiency may be increased to 45 per cent., but, as previously observed, this is seldom if ever practicable in mining, so that about 30 per cent. seems to be as high as can reasonably be expected in ordinary cases. With electricity a total efficiency of from 60 to 70 per cent. from I. H. P. of engine to Mech. H. P. on motor-shaft can easily be obtained. It must be remembered that in the Rouse tests the power was measured at the pump-discharge and not on the motor-shaft, thus including in the losses the friction of pump and gearing and correspondingly reducing the measured efficiency.

All of these tests appear to confirm the generally accepted opinion that for power-transmission purposes, except under peculiarly favorable conditions, compressed air cannot be considered as a competitor of electricity.

THOMAS H. LEGGETT, Johannesburg, South Africa (communication to the Secretary, January, 1897): Mr. Brown's excellent description of the additional plant of the Standard Co. has but just reached me. With reference to the general plan involved in these additions to the original plant, it may be stated that it is well understood by the writer that it is not good practice to put mine-hoists, pumps, etc., upon the same electrical circuit with a stamp-mill, more especially where concentrators, requiring unvarying speed, are employed.

But, in this case, the cost of the additional generator, motor and 13-mile wire-line rendered any other course impossible, it being necessary in mining, as in any other business, to cut one's coat in accordance with the amount of cloth available, even at the sacrifice of the best engineering practice.

While the direct-current machines are not literally upon the alternating-current circuit, they are practically so, inasmuch as they are driven by the A. C. motor by means of a tandem-belt; hence their load is thrown back directly upon this motor, and through it to the single generator at the power-house. This method of driving the D. C. generator was adopted, again for economical reasons, to avoid additional construction-work, and it was found to work admirably, even at the rather high belt-speed of 5600 feet per minute.

The throwing of this somewhat trying combination of varying loads upon a motor driving a stamp-mill and vanner concentrators was undertaken with less hesitancy because it was known that the loads could be so adjusted that they would not all fall upon the A. C. machines at the same time, and, further, that the mill-engine fly-wheel could be used in the event of too great variations of speed.

In short, while the combination in this case has been a success for the reasons given, it is undeniably better practice to keep mill- and mine-loads upon separate circuits wherever possible.

In reference to the water-power end of the line, Mr. Brown speaks of the "king-log" at the foot of the free face of the dam as bearing "to a certain extent the whole thrust of the water." Considering the manner in which the foundations of the dam were built, it would seem as if the thrust of the water were pretty equally distributed over the entire base. Trenches were dug down to the firm blue cement-gravel, lying about 7 feet below the surface, and each log was firmly bedded thereon. The entire material was not removed, but only sufficient for the trenches, thereby leaving within each crib of the lowermost or foundation timbers a compact mass, 6 to 7 feet high, of the original gravel forming the creek-bottom.

The dam was designed and built 57½ feet and not 60 feet thick at the base, against which there was probably too much water for Mr. Brown to obtain an accurate measurement. The present dam is, however, built exceptionally strong, not only for the reasons stated (cloud-bursts, freshets at time of melting snows, etc.), but largely because it was the intention to increase its height to the full limit in the near future, thereby greatly increasing the storage-capacity.

Of the pipe-line, the lower 655 feet of 18-inch pipe is of No. 12 iron (instead of No. 14), as there the pressure is, of course, the greatest.

The plant has demonstrated the adaptability of electric power to the various kinds of mining work; and now that the amalgamation of the chief mines in Bodie is finally accomplished, there is a field for the extension of the transmission.

The Magnetic Separation of Non-Magnetic Material.

Discussion of the Paper by Messrs. Wilkens and Nitze (see p. 351).

(Pittsburgh Meeting, February, 1896.)

William B. Phillips, Birmingham, Ala.: The questions raised by Messrs. Wilkens and Nitze are in the highest degree interesting to owners of low-grade iron-ores, and no less so to blast-furnace managers. If it can be shown that ores heretofore considered worthless can be brought within the limit of economic use, many large and readily accessible deposits may come into market on the one hand, while on the other the possible increase in the output of the furnace per ton of ore used will prove of advantage to the iron-maker.

It cannot, of course, be claimed that this process has been employed on a working-scale with respect to iron-ores, and in so far as concerns results on a large scale no one is any wiser than his fellow. Granting that the results obtained in an experimental way (and certainly the experiments were conducted with a great deal of care) can be attained in practice, we would still be facing the question, what it would cost to get them. In the absence of any specific data, we have to fall back on general principles, helped along, as far as possible, by indications afforded by the experiments.

I must confess that as regards the Clinton ores, which I have studied to some extent, I am not sanguine of making a ton of 53 to 54 per cent. concentrates from less than 1.75 tons of 40 per cent. raw ore. It may be possible to get the iron in the heads to 56 to 57 per cent., but the lesser richness of the middlings will probably reduce this to 53 to 54 per cent. in the salable stuff. As the authors remark, it is not worth while to discuss the concentration of the better grades of "soft" Clinton ore; for the price at which they are now delivered at the furnaces, and the comparatively limited amount now remaining, render their concentration unadvisable.

If it is to succeed in the Birmingham district, the process must be applied to the low-grade soft and hard ores now con-

sidered worthless; and if it succeeds here it will succeed anywhere.

The ore with which the concentrates will come into competition here is mined and sold at so low a figure that the successful operation of a concentrating-plant would demand the very best efforts of the most competent men. The process appears simple enough, and it is in fact so simple as to challenge the admiration of those who have had to do with other processes proposed. To dry, crush, size and send ore to a separator! No roasting, no magnetizing, no large volumes of heated gas to deal with, no sintering! It is an ideal process. The heads can be brought up to 56 to 57 per cent., and the tails down to 10 per cent. of iron, and the middlings can be distributed anywhere between these limits.

But does there not lurk somewhere the inevitable difficulty? Is it all to be plain sailing? What are the points in connection with the application of this process to the Clinton ores that will probably demand the most attention, and will, to a great degree, condition success?

These remarks must not be misinterpreted. I have kept a very close watch on the process since its application to these ores was suggested by Mr. Wilkens, and I am convinced, after many careful experiments on a considerable scale, that it is today the best process yet devised for the improvement of the low-grade Clinton ores, both soft and hard. I have urged its adoption, not from hearsay evidence of its value, but from actual experience with it. The obstacles in the way of its use are such as would be met with in any system of concentration, viz., the cost of preparing the ore for the concentrating-machine. The act of separating the ore is a minor point. The fact being once established that it can be done, the details of the operation can be worked out according to local conditions. By far the greater part of the expense is incurred between the crusher and the separator-bin, and is liable to wide fluctuations.

It is not likely that with the Clinton ores a size coarser than 0.1 inch diameter can be profitably treated. It may be that on some of the ores crushing through an 8-mesh screen would suffice; but, as a rule, somewhat finer crushing will have to be practiced.

The cost of crushing will, of course, depend, *ceteris paribus*,

upon the ultimate fineness to which the ore is to be carried. To counterbalance the increased cost of crushing through a 10-mesh screen, we must be able to effect a more thorough concentration, and to make either more heads or richer heads. That is to say, a ton of ore put through a 10-mesh screen must yield more profit, when concentrated, than a ton of the same ore put through an 8-mesh screen. The size of the meshes that all of the ore must pass will depend on the nature of the ore; and this is determined not only by the relative amounts of oxide of iron and siliceous matter present, but also, and particularly, by the relative hardness of the two. To illustrate: some of the low-grade, soft Clinton ore, on being crushed and ground to pass a 10-mesh screen, will yield from 23 to 33 per cent. through a 40-mesh screen, and this material will carry from 50 to 54 per cent. of iron without further treatment. The original ore carries from 35 to 40 per cent. of iron, with about the same amount of silica. The material left on the coarser screens carries much less iron than the fines through a 40-mesh. In this case the iron-bearing portion of the ore is much softer than the matrix; it grinds to a finer powder, and by simple screening is removed from the harder (tougher) portion, and constitutes in and for itself very good ore.

This is the case generally with the low-grade ore of Red mountain, although there are some exceptions.

But there is a limit to the beneficiation of the ore by screening. When the amount of material passing a 40-mesh screen exceeds 25 per cent., there is a tendency towards retrogression in quality. Even at 25 per cent. a falling-off is observed; and the irons in the fines will not be above 50 to 51 per cent. It follows naturally that in preparing the ore for the Wetherill machines we would have to decide how much fines there should be, and what disposition should be made of them. If the quantity can be kept below 25 per cent. we may expect, by simply screening the crushed ore over a 40-mesh screen, to get fines that will carry from 52 to 54 per cent. of iron. This material I would not attempt to concentrate further. It may be improved, it is true, but the improvement would not be worth the trouble and expense. Setting it aside, therefore, as blast-furnace-ore direct, or as material to be mixed with heads or middlings, as the case may be, or to be used as paint after

finer grinding, we would thus dispose of, let us say, 23 per cent. of the ore. What is it worth? As ore it probably would not bring as high a price as ore of the same grade, but coarser. Excluding the question of briquetting, as introducing additional expense not provided for, and looking at the matter solely from the standpoint of the furnace, it is not likely that the fines would bring, *as ore*, more than 52½ cents per ton f. o. b. at the works. In a report on the process, I have taken the value of fines carrying 49 per cent. of iron at 45 cents per ton f. o. b. at the works. If they should carry 51 per cent. of iron they might be worth the additional 7½ cents. In that report the amount of fines was taken at 32 per cent., and if there should be made only 23 per cent., we could expect to secure the additional 2 per cent. of iron.

We have, therefore, 23 per cent. of the ore worth 52½ cents per ton, and as nearly all of the expense incurred in the use of the process has to be met before the ore goes to the separator, this portion must bear its share.

What can be done with the other 77 per cent.? Of what richness can the concentrates be made, and what will they be worth? How low in iron can the tails be made, and what will be the inevitable loss from this source?

We will allow that 70 per cent. of the material passing over a 40-mesh screen can be brought up to 56 per cent. of iron; that 15 per cent. will be middlings, which cannot be carried above 50 per cent. of iron, and that 15 per cent. will be tails carrying not over 15 per cent. of iron. How will the matter then stand? From 100 tons of ore we would get, by simple screening over a 40-mesh screen, 23 per cent., or 23 tons, of fines, worth 52½ cents per ton.

Allowing that 70 per cent. of the remainder, or 54 per cent. of the original ore, can be brought up to 56 per cent. of iron, we would have 54 tons of this quality, worth, we will say, $1.00 per ton; of middlings we would have 11.5 tons, worth, say, 60 cents per ton, and 11.5 tons of tails of no value.

The account then would stand:

23 tons	@ $ .52½	$12.07
54 "	@ 1.00	54.00
11.5 "	@ .60	6.90
		$72.97

Or, say, 73 cents per ton of original ore. That is, we would get from a ton of raw, *dry* ore products worth 73 cents. This I am inclined to consider the maximum value of a ton of soft Clinton ore turned into fines and concentrates, the fines being used as ore and not for paint. The minimum value lies somewhere about 47 cents. The difference between these figures, 26 cents, represents the possibilities of the process per ton of raw, dry ore. I do not say that 26 cents per ton of raw, dry ore represents the possible profit. Calculated profits are apt to be illusory. In the absence of specific data on which to base an opinion as to the working of the process on a large scale, with Clinton ores, we shall have to wait.

That the process is unusually promising, and worthy of the closest attention, cannot be doubted by any one who has taken the trouble to look into it. The paper under discussion is sure to attract a good deal of comment, and I am fortunate in being able to add my mite to the general contribution.

R. W. Raymond, New York City: The allusion in this paper to the gap between the strongly and the weakly paramagnetic substances may be more fully illustrated by a reference to the work of Plücker, whom the authors mention in passing. His paper on the subject is in Poggendorff's *Annalen*, vol. lxxiv., for 1848. Taking the magnetic attractability of iron as 100,000, he determined that of other substances by careful experiment. The following instances, extracted from a larger number in his paper, will be interesting in this connection:

Magnetite,	40,227
Red hematite,	134
Specular iron-ore,	533
Hydrated ferric oxide,	156
Limonite,	71
Dry persulphate of iron,	111
Protosulphate (iron vitriol),	78
Pyrite,	150
Hydrated Mn_2O_3,	70
Mn_3O_4,	167

It is evident that determinations of the magnetic attractability of natural minerals must be affected by their degree of purity, and also by their chemical and physical condition. To meet these points partly, Plücker calculated this function for

the amount of iron, manganese, etc., present in the minerals examined, obtaining the following figures:

Iron in magnetite,	55,552
" red hematite,	191
" specular iron ore,	761
" hydrated ferric oxide,	296
" dry persulphate,	349
" protosulphate,	385
" pyrite,	321
Manganese in hydrated Mn_2O_3,	112
" Mn_3O_4,	232

He found also a great difference between two forms of anhydrous ferric oxide (the second, I infer, being colloidal), of which one showed a susceptibility of 500, and the other of 286, the figures for the metallic iron in them being respectively 714 and 409.

His results do not agree in all respects with those of Wiedemann, or with the results of experiment by the Wetherill machine. A striking instance of discrepancy is furnished by pyrite and iron vitriol, the former of which Plücker rates at 150 and the latter at 98. Yet I was able, with the Wetherill machine, to take up practically the whole of a charge of powdered iron vitriol, whereas it has not yet been found possible to effect with that machine any extraction of pyrite. Very possibly some adjustment of quantity and intensity of current, in connection with the machine, may hereafter be found, by which it will be made effective upon a wider range of materials than has been successfully treated so far. Meanwhile, I think I may venture to say that at the present stage of this apparatus and its manipulation, the results obtained by it do not exactly agree with any of the laboratory-lists of relative attractability heretofore published by scientific investigators. The causes of discrepancy will furnish an interesting field for further inquiry.

It is pretty certain, however, that the ultimate range of the apparatus will be confined wholly to paramagnetics, and none of the diamagnetics will be added to that list. Under this latter head, Faraday included bismuth, antimony, zinc, cadmium, sodium, mercury, lead, silver, copper, gold, arsenic, uranium, rhodium, iridium and tungsten; while he placed among the paramagnetics iron, nickel, cobalt, manganese, chromium, cerium,

titanium, palladium, platinum and osmium. Some of these latter are doubtfully classified. Messrs. Wilkens and Nitze have not clearly succeeded in attracting, thus far, any minerals not containing either iron or manganese. What they may accomplish hereafter among the other paramagnetics will be watched with interest.

A Modern Silver-Lead Smelting-Plant.

Discussion of the Paper of Mr. L. S. Austin (see p. 388).

(Colorado Meeting, September, 1896.)

HENRY A. VEZIN, Denver, Colo. (communication to the Secretary, February, 1897): I have read Mr. Austin's paper with considerable interest, more especially as the designing and study of such works have been an attractive occupation for me during the last twenty-five years.

Mr. Austin says that a "plant on a terraced site has generally been considered the most advantageous," and also that "many advocate a level site," while he himself "is disposed to advocate a modification of the latter plan, utilizing an extended surface with a moderate slope."*

Among the advocates of the terraced sites as against level ones, I do not recollect ever meeting a metallurgist who had even made an attempt to plan his works for level ground, or who had, in a lucid interval, figured *how much* he gained or lost by one or the other method. The average metallurgist starts with the assumption that it is rational to conduct his material down hill because its movement is aided by gravity, while to elevate ore or furnace-material by machinery he vaguely believes to be not alone wrong in principle, but also unnecessarily expensive.

Let us look into the expense of elevating and see whether it really is prohibitory of the successful operation of smelting-works on a level site. Hoisting or elevating material in the

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* In the final form of his paper this expression of preference has been omitted. My remarks on the subject, therefore, no longer apply to Mr. Austin; but they are retained as an answer to other metallurgists and mill-men, who have often expressed to me the same preference, not liking to give up terraces altogether.

West, even with a consumption of coal as high as 10 pounds per hour per horse-power, can be readily performed for ¼ cent per ton, the lift being 30 feet. Add ¼ cent for repairs, and we find the total cost per ton ½ cent. Mr. Austin admits for works on a level site good ventilation, accessibility and compactness of plant. Now assuming, for the sake of argument, that the cost of elevating is four times as great as my estimate, is not the one point, *good ventilation*, cheap at such a price?

It is, however, unfortunate for the terrace-advocates that the cost of elevating is not saved in their system, which involves tramming much further (usually by man-power), a great amount of shovel- and barrow-work and less convenience in the general arrangements, as well as interest on the greater first cost of the works.

As to "liability to accidents," if elevators are properly constructed they are as safe as any other machinery used in smelting-works. Like the blowing-engines, they should be in duplicate when they fulfil an important office.

The advantages of a flat site over one in terraces are: (1) that the first cost of the works is smaller; (2) that the arrangement can be made more convenient, as the lay of the ground does not compel placing the different buildings or departments in a certain predestined order so as to obtain the fall required; (3) that every square foot of the ground may be at will alternatively the equivalent of an inferior or a superior terrace to every other, and hence parts of the works that, on a terraced site, must be far apart in vertical distance, can be placed on a level site side by side.

In the terrace-system the ore can only go down hill unless elevators are used; and if these are required it makes little difference whether a few more are put in or the indispensable ones are built higher. The objection has been urged that elevators, especially the platform-hoists, get out of order or break down. In cases that came under my notice I found that the management had tried to get for $300 to $400 machinery that could not be properly made for less than twice that sum. There is no reason why elevators should not run day-in-day-out with far less chance of interruption than a steam-engine, which is the most delicate machine used in mines, mills and smelting-works, and the most liable to interruption for adjustment or repairs.

I assume, of course, that the other machinery is amply strong and well-made.

I may cite a few examples of the manner in which professional men deceive themselves by not asking themselves "How much?" In the autumn of 1882, during the Colorado meeting of the Institute, a very able geologist, perfectly familiar with Leadville and its surroundings, while passing the smelting-works below the town, pointed out to the other visitors the arrangement of the plants, and wound up with the words: "as if intended by nature as the site for lead-smelting-works." I had known the arrangement of these works since 1879, or practically since most of them were built. There was always a great deal of shoveling, and much of the ore was wheeled by hand *up-hill* and shoveled *up* into the bins. It would have been easy to construct works on level ground for less money and to run them at much less cost per ton, even if it had been necessary to elevate the slag to get rid of it.

General von Helmersen, Geologist-in-Chief of Russia, who published, in 1872, a very complete geological map of the coal-fields of the Donetz in Southern Russia, spoke of the site of the iron-furnaces at Lissitchansk, in the northwestern corner of these fields, as specially designed by Providence for iron-works. His reason was that they could be arranged "rationally," *i.e.*, in terraces. As I remember the ground, it has a slope of about 15°, possibly less, and the fall from the top of ground (or furnace) to the river is 280 feet. No iron-master that I know would choose this site, even if coking coal and good ore were found in the neighborhood, which is not the case.

In 1888 the advising engineer of a wealthy syndicate hunted all over Montana in search of a site having 200 feet fall, on which copper-smelting-works were to be built. It would have been very easy, with a flat site, to give this gentleman the equivalent of 2000 feet fall, or more, if necessary, and he could then choose a place which would offer the greatest advantages as to railway-connections and power. The latter consideration probably governed the final choice, as the works were built on ground having, as I understand, an inclination of less than 3°, while the power is furnished by a river flowing past the site.

It is not probable that advocates of the two systems will ever

come to an understanding as to what they really disagree about, until each has designed works according to his ideas of what is best, and then has shown the advantages that he gains, not in uncertain terms, but in the precise language of dollars and cents per ton of ore. Heretofore, whenever I have been drawn into a discussion of the subject, I have not been able to elicit from the advocates of the terrace-system anything but vague declarations of general principles, or the assertion that the "rational" way to build works is in terraces.

Mr. Austin gives no reason for his preference for a "moderate" slope. If he expects to secure the advantages of both systems without the inconvenience of either, he has my sympathy; for to me it appears he will have the disadvantages of both. On the one hand, he loses the freedom of arranging his works to the best advantage, while, on the other, he must use, so far as I can see, as many, if not more elevators than on a level site.

Sampling.—No one, I presume, will dispute the correctness of the seven requirements for satisfactory automatic sampling, as laid down by Mr. Austin (see page 395). But after reading what he says, we are no nearer knowing how he intends to solve the problem than before. A strong prejudice exists against the use of automatic samplers amongst those who have either tried defective ones or else have misapplied good ones. One objection that I have heard urged by an able metallurgist against a mechanical sampler is that, when the stream of ore is interrupted the machine takes no sample. This is correct; but when no ore is passing no sample should be taken. The machine alluded to I sketched out about twenty-five years ago; but it was never applied until within the last few years. It consists of one or more scoops attached to a vertical revolving shaft. In passing through the stream of ore the scoops cut out an equal portion from each part of it, and thus assure a correct sample of the whole. These scoops can be arranged to take a cut out of the stream every second, or less, if necessary, and to take any proportion up to one-half of the ore. As regards the continuous stream of ore, there should be no difficulty in maintaining a uniform stream, fluctuating only proportionally to the rate of feed to the crushing-machinery. But assuming that the stream, through carelessness or design, is intermittent, the chances are entirely in favor of the sample

being correct if taken every second. Not the least advantage of this sampler is the fact that it is not hampered by any patent-rights.

Another objection urged against a good sampling-machine is that the coarse pieces of ore which strike the edge of the scoop are assumed to jump off and go with the rejected portion of the ore—that is, most of them do so, and hence the sample does not contain its due proportion of lumps, and may be richer than the average of the ore. To remedy this alleged fault, I propose having the rejected portion caught by the scoop and the part that falls outside taken as the sample. For this purpose the scoop is made as large as necessary to take the proportion ($\frac{3}{4}$, $\frac{4}{5}$, $\frac{5}{6}$, $\frac{7}{8}$, etc.) of the stream that is to be rejected. This arrangement should satisfy the most anxious metallurgist.

In several cases the machine above alluded to has been arranged to take duplicate samples, so that they might check each other. One machine took $\frac{1}{16}$ of the stream once every six seconds; the other, the same amount just as often, but three seconds later, or half-way between the scoopfuls of the first. The samples checked so closely that the method was soon abandoned, and the two sixteenths were run together. If the resulting sample is so large that it can safely be cut down again, the largest pieces being $1\frac{1}{2}$ inches in diameter, it is shoveled into the crusher again, so as to pass as a continuous stream over the sampler. If this takes $\frac{1}{8}$, the second sample is $\frac{1}{64}$ of the original lot. This proportion has furnished perfect samples in treating Cripple Creek ores containing 1 to 2 ounces of gold per ton. I think, however, that rather more should be taken, and more frequently, especially if duplicate samples are not taken. With richer ores, especially when the values are very unevenly distributed, it would be best to take not less than $\frac{1}{5}$, and take a slice out of the stream not less than once every 1 or 2 seconds.

As to cleaning the sampling-machinery, the boot of the elevator and sometimes the spouts need it. The sampling-machine cleans itself.

When small lots of ore are to be sampled, especially if they are rich, the method of "coning and quartering," sanctified by long usage, is employed. Assuming that the greatest care is taken to keep the apex always vertically above the point of

its first beginning, to have the distribution as nearly regular as possible, and to do the quartering by sheet-metal plates, so that, when a quarter is removed, the coarse pieces from the adjoining quarters do not fall upon the ore that is being removed, the result is still not always perfectly satisfactory. If the cutting-down is done by taking every alternate shovel for the sample, reducing the pile formed by these in the same way, a closer sample is the result, with one-third to one-fourth the amount of labor. In this way ore is treated as a stream from which a foot is taken, and the next foot rejected. If the pile contains 4 tons, and a shovelful weighs 10 pounds, as might be the case with heavy ore, 800 cuts are taken. This sample must certainly be closer than when, in quartering, only 4 cuts are made, for the supposed mixing due to coning cannot possibly compensate for the error due to so few cuts. The sampling of small lots, or the cutting-down of the samples from the main automatic sampler, can be done by one man, if he has a small platform-hoist at his disposal. He would hoist the ore in a car, or in suitable cans on wheels, and dump it over a good divider. From this the ore would fall into similar cars or cans, and the operation would be repeated until the required reduction was obtained. There would be no sweeping-up or shoveling. When it became necessary to crush the ore finer, the same hoist would serve to raise it so that it could be dumped into the hopper over the feeding-shoe of the sampling-rolls. The crushed ore from the latter would be received in a pan on wheels or a car. There would be no feeding-arrangement for rolls, consisting of a man shaking a shovel. Thus, one man could do the work of at least eight, and more satisfactorily. The sampling-rolls should be very strong, so as to crush. I recently saw a pair, 20 by 12 inches, that could safely be set for a pressure of 20 to 25 tons between the rolls.

Mr. Austin (page 397) speaks of the water-jackets, whether of cast-iron or wrought-iron, as unsatisfactory. The life of a cast-iron water-jacket, if made of the best iron, cast in dry sand, is three years and upwards. According to the repairs during the past year, the average life of a jacket in one of the smelting-works of Denver is more nearly ten years. The loss of first-class cast jackets is now almost always the consequence of carelessness on the part of the attendant. The cost per

pound is 3 cents, making the cost of a jacket, 20 inches wide and weighing 450 pounds, $13.50. If made of sheet-metal, its weight is about half as great; but it costs about three times as much if 20 inches wide, and fully twice as much if each side of the furnace is made of two jackets. I myself know but two works where steel jackets were used. In one of these, each side of the furnace consisted of one jacket. They gave a good deal of trouble in the beginning, due, to some extent, to the lack of skill on the part of the maker. In the other works, each side was formed by two jackets. These were very well made and carefully handled; but within four years some of them had to be patched, increasing very much the liability to burn out or become leaky. In 1881 or 1882 three or four months were considered a good life for a cast-iron jacket. They were then cast in green sand, and cost about 7 cents a pound in Leadville. Some economical metallurgist beat the foundry down ¼ cent per pound, and reduced the quality in consequence. Poorer iron was used; and the jackets lasted, at most, six weeks. At certain works in Colorado, as recently as four or five years ago, cast jackets made in green sand by men who evidently didn't know how, broke, when the furnace was blown in, before it was half full; and none of them lasted over a week or two. Hence the growth of feeling in favor of the wrought jacket. The life of a well-made cast jacket, its low cost and the ease with which it can be replaced, hardly justify Mr. Austin's expression.

Air-Leaks. (Page 398).—The principal loss is probably through the infinite number of fine holes in the canvas sacks or hose. They can be made air-tight by coating the inside with a thin covering of glycerine glue, which remains pliable like rubber.

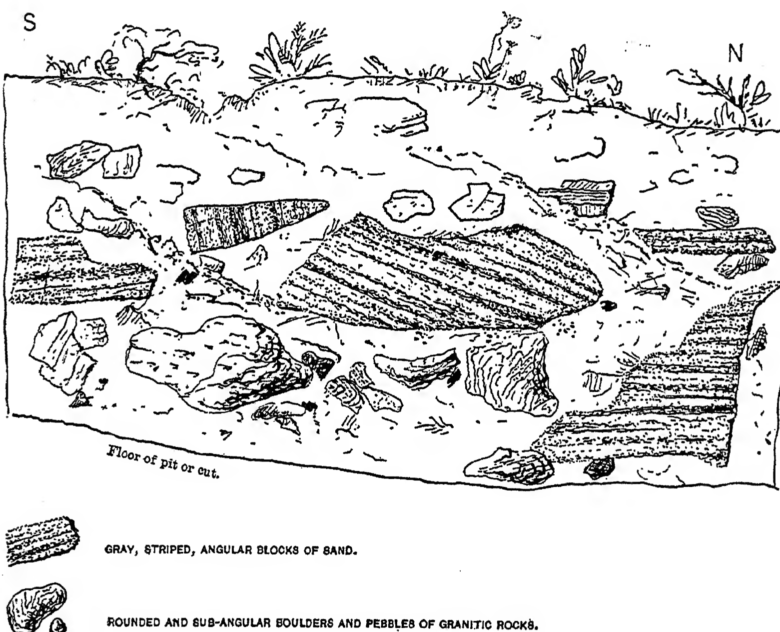
Blowers. (Page 398).—I think Mr. Austin will find that the cost of a cylinder-blower is but 20 to 25 per cent. greater than that of a rotary pressure-blower and engine to drive it, the two having the same capacity according to the catalogues. The capacity of the latter is given as the displacement of the impellers without allowance for back-leak or slip. With pressures from 1 to 2 pounds, this slip becomes very great; and it is probable that for the same actual capacity at 2 pounds pressure the cylinder-blower will cost no more, if as much, as the rotary.

Faulting in Glacial Gravel.

Discussion of the Paper of Mr. Carl Henrich (see p. 460).

(Colorado Meeting, September, 1896.)

W. S. Gresley, Erie, Pa.: Mr. Henrich's Fig. 1 and reference thereto remind me of a section of sand, gravel, clay, etc., which I sketched near Hellesylt on Stor Fjord, Norway, some years ago.



Vertical Section of Glacial Morainic Material, containing Angular Masses of Banded Sand. Observed June 29, 1887, about 1½ English miles south of Hellesylt, at head of Stor Fjord, near Aalesund, Norway.

The accompanying figure reproduces my sketch. It was the angular and tilted blocks of *sand* in this glacial *débris*, or moraine, that particularly attracted my notice. The sand was moist; and I could only account for the phenomenon of *blocks of sand*

embedded in a heterogeneous mass of boulders, pebbles, clay, etc., by supposing that the bed of sand which furnished the blocks of it in the moraine was *frozen solid* at the time the fragments were broken off it, transported and buried in the general *dêbris*. The idea that the masses of sand had once been sandstone, and in that condition had been broken and dumped and buried in the morainic material, could not be entertained, because there is probably no sandstone in the district, but there was a good deal of the same sand in natural position which I observed within a few rods of the section. Since then I have seen near Blossburg in Tioga county, Pa., a section of coarse gravel about 30 feet high, through which from top to bottom (as far as the exposure went) ran a nearly vertical vein of fine yellow sand. In this case, also, it occurred to me that the gravel-bed was probably in a frozen condition at the time it became split open and received the vein of sand.

May not the faults in Mr. Henrich's "Kelly's Sand- and Gravel-Cut" have been produced by pressure upon the beds *when frozen hard*, and therefore, presumably, in a fit condition for breakage and dislocation as observed?

The Occurrence and Behavior of Tellurium in Gold-Ores, More Particularly With Reference to the Potsdam Ores of the Black Hills, South Dakota.

Discussion of the Paper of Mr. Frank Clemes Smith (see p. 485).

(Colorado Meeting, September, 1896.)

Walter P. Jenney, Rapid City, S. Dak.: This paper is the first publication, I believe, of the fact that traces, up to two or three ounces to the ton, of tellurium have been found in the gold ores of the Black Hills occurring in the Potsdam formation.

There are two great classes of ore-deposits in the Black Hills. First, the gold-ores in the Archæan, of which the great Homestake lode, extending from Lead City to Central, in the northwest section of the Hills, may be taken as a type; second, the refractory gold-ores found in the Potsdam sandstone over a

great area surrounding Terry's Peak and Bald Mountain, about 8 miles southwesterly from the city of Deadwood. These refractory ores are peculiar in that the rock is slightly mineralized Potsdam sandstone. In the unoxidized portions small quantities of iron pyrites occur disseminated in minute particles through the rock. In the oxidized portions, all the iron appears to occur in the form of peroxide. Strangely enough, there has been, until this time, no evidence whatever as to what form the gold occurs in. We have never been able to find a piece of ore so rich that, under the microscope, the mineral form in which the gold occurred could be determined. All that was known was that the very richest ores contained from 22 to 27 per cent. of their gold free, so that it could be separated in the form of a fine, brownish powder on panning. This free gold will amalgamate; but the greater portion of the gold contained in the ore resists amalgamation even when carried out in steam-heated pans. The discovery is announced in this paper of a small percentage of tellurium in these refractory ores, so that it is not improbable that the gold may occur in some form of telluride, though we have not as yet been able to obtain the gold-bearing mineral sufficiently pure, so that it might be subjected to analysis and the true mineral composition determined.

The district is very important from a mining standpoint. Last year, including the production from the Homestake lode, the northeast section of the Black Hills produced more than four million dollars in gold, and about one hundred and sixty thousand ounces of silver. This year, owing to the extending of the work of prospecting on the refractory belt, and the enlargement of the Homestake mills, the production of the district will probably exceed five million dollars; so that this region in the future promises to contend for first or second place with Cripple Creek in the production of gold in the United States. It has this great advantage: the area has two undetermined boundaries, on the south and on the west.

The Potsdam formation, still carrying workable bodies of refractory gold-ore, dips under Carboniferous limestone, forming the main divide of the Black Hills. The dip is very shallow, frequently not more than 20 feet to the mile, but in some instances as much as 160 feet to the mile, so that it is practicable to follow these ore-deposits underneath the limestone by shafts

sunk through the overlying formation. As at present explored, the district measures about 8 miles northerly and southerly by 6 miles easterly and westerly, in its greatest extension.

These refractory ore-deposits in the Potsdam are all fissure-formed; that is, the ore has been introduced through a vertical fissure, traversing the nearly horizontal layers of the sandstone, and has been deposited on one or both sides of the fissure in certain favorable layers of the rock.

It is a district well worthy of the attention of the Institute and should be brought to the attention of the United States Geological Survey, that the formation and peculiar occurrence of these ore-deposits may be carefully worked out. It is a district of which very little is known, and its study cannot fail to yield a great deal of information regarding the formation of ore-bodies, especially that class of ore-bodies in which the ore, introduced into the rocks through fissures, lies almost wholly outside of the walls of the crevices by which it entered.

I have not made a special investigation of the action of tellurium in the assay of gold-ores, but it is not improbable that the losses in assay resulting from the presence of tellurium may be analogous to those produced in silver-ores where large quantities of antimony are present in the sample. In investigating the rich antimonial silver-ores of the Wood River district, Idaho, it was discovered that the losses due to antimony could be entirely prevented by the use of an oxidizing charge, so that all the antimony passed into the slag and the button resulting was pure lead, containing practically all the silver contained in the ore. This was effected by taking an extremely small charge of ore,—from one-twentieth to one one-hundredth of an assay-ton; the latter being the charge for ores carrying 2000 ounces of silver per ton. Crucible-assays alone were made; the charge containing a great excess of litharge with the addition of from 2 to 5 grammes of niter, so that the resulting button would weigh from 5 to 10 grammes. The lead buttons, readily separated from the slag, were bright, soft and malleable, and on cupellation gave results which closely checked. It certainly would be worth while to experiment on the assay of tellurium-ores in this way; that is, to make an oxidizing charge, so as to keep the tellurium out of the button, fluxing it all into the slag, and thus obtaining a button of pure lead, carrying the gold, which could be cupelled with the least possible loss.

H. VAN F. FURMAN, Denver, Colo.: Dr. Jenney is in error when he says that this paper is the first publication of the discovery of tellurium in the Potsdam ores of the Black Hills; and perhaps he has been misled by the omission of Mr. Smith to mention clearly the circumstances of that discovery. The paper, however, does allude (though without giving dates) to a prior publication by Mr. Richard Pearce, in a paper on "The Occurrence of Tellurium in Oxidized Form Associated with Gold," read April 1, 1895, before the Colorado Scientific Society, at Denver. It appears from that paper that Mr. Smith himself sent to Mr. Pearce, "for the purpose of determining to what extent the mineral could be roasted preliminary to chlorination," the samples in which tellurium was detected, and that he afterwards forwarded, at Mr. Pearce's suggestion, picked specimens of the blue and red varieties, for the quantitative determination of the tellurium existing unoxidized in the former and oxidized in the latter. Some of these specimens are now in the mineral collection of the Colorado Scientific Society. Chemical analysis demonstrated the presence of tellurium in both varieties, while careful vanning failed to show the presence of free gold. Mr. Pearce ventured the opinion that the gold-bearing rock was an altered phonolite, and called attention to the analogy between these ores and those of the Cripple Creek district in Colorado. He gave also the results of some experiments in roasting telluride-ores. The amount of tellurium lost in roasting was small in each case, but this loss was greatest when pyrites was present. The tellurium remained behind in the form of TeO_2, either free or combined with ferric oxide. These experiments led Mr. Pearce to the conclusion that the volatilization of tellurium and accompanying loss of gold in such roasting were not so great as had been supposed.

In his Presidential Address of February, 1890,* Mr. Pearce pointed out the occurrence of tellurium in the ores of Gilpin county and of Leadville. In a later communication to the Colorado Scientific Society, he reports its occurrence in the ore of the Mayflower mine, Butte, Montana.

With regard to the assay of these telluride-ores, Mr. Smith says, in the first place, that he pulverized them, and passed them through a 60-mesh screen. Now all of us here in Colo-

* *Trans.*, xviii., 447.

rado who have had anything to do with the assay of tellurium-ores (and we have had a good deal in the last 3 or 4 years), know that a sample crushed through a 60-mesh screen cannot represent the ore from which it was taken. I think many of the differences in assay that Mr. Smith records may be due to the fact that the pulp-sample for assay was not sufficiently pulverized. The practice in Colorado, at the present time, is to pass all pulp-samples of high-grade ores, and especially of tellurium-ores, through at least a 100-mesh sieve; and I think many of the smelters and sampling-works now-a-days use a 120-mesh, which is certainly a step in the right direction.

Mr. Brunton's paper,* read at the Atlanta meeting of the Institute, October, 1895, calls attention to this matter, and proves conclusively by mathematical analysis the necessity of finely pulverizing these ores. This applies not only to the final pulp-sample taken for assay, but often to the whole original sample before quartering. At many of our smelting-works it has been found, in the case of tellurium-ores like those of Cripple Creek, where the gold is intimately associated with the andesitic or other gangue, forming a mass far from uniform in composition, that it is necessary to crush the whole sample of ore fine before the small sample is finally cut out. I believe that many of our smelters now crush those ores to a quarter-mesh, at least, before they attempt to sample them. Again, the majority of our works, I believe, have come to the conclusion that the best method is by coning and quartering, or hand-sampling.

As Mr. Jenney has said, the use of a large amount of litharge in the assay of these ores is absolutely essential. I regard these tellurium-ores as really the easiest gold-ores to assay that we have in Colorado. There should be no trouble in assaying tellurides. It is a simple matter, either by scorification or by crucible. There are two points to be observed in crucible-assays: there must be a large excess of litharge present, and the crucible should be introduced with the furnace at a low temperature, and the furnace gradually heated up. It will not do to put these ores into the furnace at a very high temperature. My own experience is, that the fusion of ordinary ores should require about 20 minutes, while these ores should require for fusion about 30 minutes. There is no difficulty in the scorifi-

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* *Trans.*, xxv., 826.

cation-assay. The only point to be observed is to add a little litharge (generally 10 or 15 grammes), either mixed with the ore or as a cover, and then start scorification at a very low temperature, and gradually bring the muffle up to a high heat. I think that is the whole secret of the assay of these ores.

Some time ago I made a large number of experiments on the assay of some very rich telluride-ores in the Independence mine at Cripple Creek. I had several different samples, running all the way from 9 to 39 ounces of gold to the ton. When I received these samples I carefully divided them, and sent one-half of each sample to a chemist in the east, and asked him to determine the gold by chemical analysis, and, in the meantime, I went to work and assayed the ores by both the ordinary crucible- and the scorification-assay. There was scarcely any difference between the results as obtained by chemical analysis and my results by fire-assay. The average of all my results was a trifle higher in gold than the chemical analysis showed, which might fairly be attributed to the small amount of silver left in the assay after parting.

Concentration of Ores in the Butte District.

Discussion of the Paper of Mr. C. W. Goodale (see p. 599).

(Colorado Meeting, September, 1896.)

Robert H. Richards, Boston, Mass.: Mr. Goodale calls attention to the difficulty of obtaining clean tailings from the jigs which are fed by the spigots of the hydraulic separator. This difficulty is universally encountered. The Lake Superior copper mills practically overcome it by running the beds of those jigs loose, carrying into the hutch all of the available copper and some sand. These hutches are again jigged upon three-sieve Collom jigs, where the treatment is very slow and gentle; and on the successive jigs lighter and lighter bedding is used. The last sieve is run thin enough to let some sand into the hutch. With these precautions most of the free copper is extracted, as well as a fair proportion of the included copper.

In comparing the vanner and the conical slime-table, Mr. Goodale does not tell us how the latter was fed. No table can compete with a vanner unless it is fed with water-sorted material, in which the particles of ore are smaller than those of the quartz with which it is mixed. I have on several occasions heard the table condemned, and upon questioning I found that the whole pulp from a stamp-mill was sent to it without any classification. The table was asked to do work for which none of its best friends would advise its use. Perhaps Mr. Goodale has looked out for this. If so, the result is certainly very surprising.

The sizing and assaying of the vanner-tailings are very interesting and suggestive. The coarser sizes run high, presumably from included grains; the finer run high from free ore, a portion only of which the vanner can save—a much larger proportion of which one would expect the table to save.

If Mr. Goodale's samples were taken from the whole launder periodically for the whole twenty-four hours, then his results must be final for the ore in question; but if the sample was taken with a hand-dipper from under the tail-roll of the vanner, and only during the day, then it is quite proper to question whether he has proved that a guarding-machine, like a table, will not pay for itself.

Another way to come at the question is to put the unclassed or unsorted pulp directly upon a table, and take out the fines and send the tailings from this treatment to the vanners.

Mr. Goodale's tests show the edges of the vanner to be decidedly richer than the middle part. This is an important point proved, and one on which the designers of new vanners should keep watch. For example, the Tulloch concentrator has to the eye killed the banks of the Frue, but what it has saved in that way it has more or less lost by the quick current on the edges. I have found that the edges invariably show on the vanning-shovel more ore than the center.

Mr. Goodale's test of vanner-treatment of a series of hydraulic-classifier spigots on successive vanners is very interesting and suggestive. May it not be possible that if, after giving the earlier coarser spigots to the successive vanners, he had given the latter to suitable slime-tables or blanket-tables, he would have obtained a more perfect result?

MR. GOODALE: Referring to the points raised by Prof. Richards, I would say:

1. *Finishing-Jigs for the Hutch from First Jigging of Water-Sized Material.*—It will be seen from the description of the Anaconda and of the Butte Reduction Works mills, which were designed on the lines of Lake Superior ore-dressing, that the hutch from the jigs working on the finer classes from the hydraulic sizers is rejigged or finished, and this would seem to be a practice which should commend itself to the other mills of Butte.

It is doubtful, however, if this treatment would work as well on the Butte ores as on those of Lake Superior, since it would be almost impossible to draw down into the hutch the finest and best portion of the jigged material. Comparing the gangue of the ores in the two districts, there is probably very little difference in the specific gravity; but in Lake Superior ore-dressing the mineral to be saved has a uniform gravity of 8.8, while in the concentrating-operations of Butte the silver- and copper-bearing minerals range from zinc-blende (about 4), through chalcopyrite, enargite, bornite, to copper-glance (5.5 to 5.8), the heaviest mineral in the mixture.

By referring to the sifting-tests on jig-tailings it will be seen that from 8 to 10 per cent. is fine enough to pass through an 80-mesh sieve, and if water enough is used on the jig to carry away the quartz-sand, some, at least, of this rich ore will go with it.

2. *Frue Vanners versus Conical Slime-Tables.*—In the comparison between the two machines in Part I. of this paper, they were furnished with the same class of material, and were running side by side in the same mill; but in the test recorded in Part II. the vanner-work in the Colorado concentrator was compared with the table-results in the Butte Reduction Works, the ore received by the two mills being from the same mine and of the same character and assay. The ore going upon the tables was classified, as shown in the description of the Butte Reduction Works.

3. *Method of Sampling Vanner-Tailings.*—The tailings-samples from a lot of 3400 tons of ore, referred to as having been tested by Mr. Pearce for the purpose of determining whether it would be profitable to attempt to re-treat such material, were taken from the whole launder periodically during the time required to run this lot through the mill.

A guarding-machine, as a follower for a vanner, would doubtless play an important part in mill-work where the valuable minerals escaping are of high grade. In the California gold-mills, for instance, it has been found profitable to work the vanner-tailings on canvas tables, but in this case the "catch" contains material of much higher grade than the minerals which escape from vanners treating Butte ores.

4. "*Banks*" *on Vanner-Belts.*—An effort was made in one of the Butte mills to prevent the drifting of sand to the edges of the vanner-belt by suspending deflectors made of thin wood, shod with rubber, from the frames of the machine at points where the accumulation of pulp was greatest. This device was patented, but it failed to accomplish the desired results.

Referring to the Tulloch concentrator, and its loss by quick currents on the edges of what it gains by preventing the formation of "banks," I would say that the work of the Tulloch and Frue has been compared in a Butte mill which treats custom-ores, and the results were in all cases in favor of the Frue. On seven lots the tailings from the Tulloch contained 0.3 ounce silver and 0.3 per cent. copper more than those from the Frue, the "feed" of the two machines being the same in quality and quantity.

The adjustments of the two machines were as follows:

	Vanners.	Tullochs.
Revolutions per minute,	196	160
Grade of belt,	3½ in. in 12 ft.	3½ in. in 12 ft.
Belt-travel per minute,	40 in.	40 in.

John Carkeek, Butte, Mont.: Mr. Goodale calls attention to losses unaccounted for on a lot of Parrot ore worked. This is no unusual occurrence, and is due to the fact that the methods of sampling tailings are not always careful and thorough. I think that perhaps if a sample were properly taken from the tail-race, say every hour during the run, a good deal of this loss would be accounted for. To take such a sample the whole stream running through the race, water and tailings, should be diverted into a tank (not allowing it to overflow) and allowed to settle until it is time to take the next sample, say an hour. Then all the clean water should be drawn off through plugs or by syphon, and the tank filled from the tail-race again as before,

this being repeated every hour. At the end of every shift the entire contents of the tank should be taken as a sample for assay, after either filtering or evaporating the water. The floating material, which is frequently rejected in sampling tailings, cannot be saved with the present appliances in Butte, and I doubt, with present price of labor and difficulty of saving it by a settling system, whether it would pay for capital invested; but in determining losses in tailings it should be included in the sampling.

Mr. Goodale calls attention to the imperfections of the hydraulic separators, and the consequent loss in tailings from jigs fed by them. I have used a good many kinds of separators, but have never yet found a perfect one. If tailings from hydraulic separator-jigs are vanned on a vanning-shovel they will almost invariably show a very good "edge" of fine ore, which should go to slime-machines; but if water enough is used in the separators to send this to slime-tanks it throws over the coarser particles of silica and included material, which are too coarse for tables or vanners. It also gives an excess of water to slime-settling tanks; and here a difficulty is encountered in the overflow from these tanks which carries off in the current the very material we strive to save—that is, the very fine particles of ore that will float if water is excessively used. Professor Richards in his remarks describes the method adopted by the Lake Superior mills for overcoming the losses from jigs handling hydraulic-separator products. This practice, if employed on Butte ores, would, I think, result in a greater loss than results from making a final product in one operation. In the Lake Superior ores the yield in concentrates is only 1 ton from 40 to 50 tons of crude ore, while in Butte the product is 1 from 2 or 3.

It very frequently occurs that 50 per cent. of the weight of the crude ore is sent to the smelter as good clean concentrates, and to re-treat this amount means loss, therefore a separation should be effected with the least possible handling. By using a five-compartment jig a good clean separation can be made on the first three compartments; middlings (which would be for the most part included material and some sand) can be got in the hutch of the last two compartments, and good clean tailings at the overflow, the middlings to be recrushed and worked over

jigs, tables and vanners. Very good work can be done in this way, and with less loss, I think, than if the whole hutch-work were subjected to the second treatment.

The "sliming" of Butte copper-silver ores can be obviated to a great extent by a coarser system of jigging, that is, by jigging everything possible after it goes through the rock-breaker.

During the last two years I have obtained very satisfactory results on such ores by jigging material that passes through a 2⅛-inch round hole for the coarsest size, 1⅜-inch round hole for the second size, ⅗-inch round hole for the third size and so on down to 2½ millimeters (see description of Butte and Boston concentrator in Mr. Goodale's paper). The separation and jigging of the 2⅛- and 1⅜-inch material, before crushing in rolls, has the effect of reducing the amount of fine sulphides which would otherwise pass on to the hydraulic separators, and consequently the loss of which Mr. Goodale speaks is diminished. We should bear in mind, however, that not all the fines and slimes are made in the mills, but some are made before the ore comes to the mill, and, as Mr. Goodale rightly remarks, not all the material that is fine enough for treatment on concentrating-machines passes through the rolls in modern Butte mills.

In comparing conical tables and vanners, if Mr. Goodale had made clean concentrates, a fair amount of middlings, and clean tailings, on the table, and had run the middlings on the Frue vanners, the result, perhaps, might have been different. In this way I think the conical table should have a Frue vanner as its guard.

In regard to the classification of slimes for vanner-work on Butte ores, I find that by settling in the feed-tanks classification is effected to a certain extent, but better results can be obtained by feeding coarse and fine together upon the machines than by working them separately—providing, as Mr. Goodale says, that no material goes to the vanner which is too coarse for it to handle and which should therefore go to the jigs.

I have found that jigs fed by hydraulic separators work the same way on Butte ores, and have frequently had to feed a little coarser material in them to keep the bed soft and even; if it accumulates on the screen I use the side-discharge frequently, or skim the sieve at intervals with a shovel.

BERNARD MACDONALD, Butte, Mont.: Mr. Goodale's paper

contains a clear, comprehensive and complete description of its subject, and adds to the literature of metallurgy a very important chapter in which the perfection and imperfections in the processes now used in the Butte concentrators are described with equal impartiality. No one can fail to feel that the data furnished by this paper have been secured by an enormous amount of painstaking labor, and that it is an absolutely reliable statement of the results obtained. As such, it affords an excellent opportunity to the members of the Institute for criticism of the methods employed, and the suggestion of such improvements as appear feasible.

Under the heading "Classification of Slimes for Vanner Work," Mr. Goodale says (p. 635):

"In one of the Butte mills, where seven Frue vanners were in use, the V-shaped feed-tanks were arranged so as to classify the material for the vanners, and the results on one lot of silver copper ore were as follows:

1. Launder carrying all slimes from mill to (2).
2. V-shaped feed-tank; settlings to (3); overflow to (4).
3. Four Frue vanners; tailings, 1 ounce silver; 1 per cent. copper.
4. V-shaped feed-tank; settlings to (5); overflow to (6).
5. No. 5 vanner; tailings, 0.64 ounce silver; 0.9 per cent. copper.
6. V-shaped feed-tank; settlings to (7); overflow to (8).
7. No. 6 vanner; tailings, 1.30 ounces silver; 1.2 per cent. copper.
8. V-shaped feed tank; settlings to (9); overflow to (10).
9. No. 7 vanner; tailings, 4 ounces silver; 2.7 per cent. copper.
10. Settling-tank; settlings, 9.2 ounces silver; 7.7 per cent. copper.

"The assays of tailings given above will show how difficult it is to save the fine sulphides, and it is the opinion of Mr. Pearce, verified by practical experience on the Butte ores, that better results are obtained on vanners when no classification is made, provided the material going upon the machine contains no grains of mineral too large for the vanner to handle properly. There is a reason for this in the effect which larger and heavier grains of material would have in entrapping the finer slimes and holding them down to the surface of the belt."

The opinion of Mr. Pearce, "that better results are obtained on vanners when no classification is made," which has been verified, according to Mr. Goodale, by practical experience on the Butte ores, surely cannot be enunciated as a proved principle in the concentration of slimes. The whole theory of concentration stands against it.

The results obtained by the seven Frue vanners on the classified slimes rather indicate the imperfection of the methods used than condemn the system. It will be seen from the quotation that the material subjected to concentration on the seven

Frue vanners was first made into four classes by settling in feed-tanks, and that the first class was fed to four of the vanners, and each of the three subsequent classes was fed to one vanner.

Before the value of such classification, as a process preliminary to vanner-concentration, can be intelligently considered, the whole amount in pounds of slimes thus classified, the quantity and character of each class and the ratio of one class to the other should be given. Without this information it cannot be ascertained for the purpose of discussion whether the duty of the vanners was too heavy or properly proportioned, according to the character of the class which each was called upon to treat. It may be said, however, that classification by settling in a feed-tank is a very primitive and imperfect method, and cannot be expected to give satisfactory results.

It would also be necessary, for the intelligent discussion of the merits or demerits of classification, to have a detailed account of the individual adjustments, and also the assays of the feed and tails, of each vanner treating a different class. If due consideration were not given to these particulars, satisfactory or intelligent results could not be expected. The principle of classification, however, should not be condemned where the manipulations necessary for the best results have been disregarded.

Strange as it may seem, the most important adjustment of vanners working on classified slimes is generally disregarded. I refer to the revolution of the crank-shaft which communicates the side-shake to the vanner. This is a mechanical motion, designed to produce on the vanner-belt the same perfect concentration as is realized on the vanning-shovel. Now, everybody realizes that in making a concentration-test of the finest slimes on the vanning-shovel a gentler and more careful motion is necessary than in making a test of the coarsest slimes. Corroborating this point, Mr. Goodale says that Mr Pearce, after making a test on the vanner-tailings of Gagnon ore, reported that "the material saved was very light and required careful handling on the vanning-shovel to collect." Why not handle the same material carefully on the vanners?

A little thought will convince any one that the number of side-shakes that would be proper for effecting the settlement

of the mineral constituents of the coarse classification of slimes on the vanner-belt, which is about 200 per minute, would be altogether too violent for producing a similar effect on the finest classifications. It would, instead, produce the opposite effect, that is, keep the fine slimes in such agitation that the metallic particles would be held in suspension and could not settle on the vanner-belt before they were carried over the tail-end of the machine.

As the motion communicated to the vanning-shovel is gentler and more carefully given when testing fine slimes, so, also, should the analogous motion in the vanner—the side-shake—be gentler (slower) when concentrating the finest slimes than when working on the coarser classifications.

I think the statement "that better results are obtained on vanners when no classification is made" still remains to be proved.

SECRETARY'S NOTE.—The discussion of the papers of Prof. Christy and others on the "Cyanide Process," and of Mr. Sauveur's paper on the "Microstructure of Steel," will be printed in volume xxvii.

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